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**ELECTRODEPOSITION
OF GOLD AND SILVER FROM
CYANIDE SOLUTIONS**

BY

S. B. CHRISTY



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ELECTRODEPOSITION OF GOLD AND SILVER FROM CYANIDE SOLUTIONS.

By S. B. CHRISTY.

INTRODUCTION.

This report on the electrodeposition of gold and silver from cyanide solutions represents work that has occupied my time at intervals during the past 20 years. The investigation has been carried on simultaneously with my duties as professor of mining and metallurgy of the University of California.

During this long investigation I have been assisted in the details of the work by a number of my former students, acting in turn as assistants. Particular mention in this connection is due W. H. Hilton, who graduated in 1900; C. T. Dozier, 1902; N. C. Stines, 1905; H. N. Herrick, 1907, and L. C. Uren, 1911.

It is impossible to publish full details of the many thousands of experiments performed. Effort has been made to present only the essential data. Wherever possible, the results of the experiments have been expressed in diagrammatic form by curves showing the relation between the simultaneous variables of the experiments.

It is necessary to show, as nearly as may be, the state of the art. This is difficult because of the meager literature on electrodeposition in the cyanide process; hence resort must be had to the specifications and claims of patents, most of which seem never to have been applied on a working scale. Such information consequently can not be taken at its face value. Expressing the essential ideas evolved in such specifications is no easy task. Clearly, in quoting from patent records it is not feasible to enumerate all of the specifications or the claims allowed the inventors. A statement of the chief contributions to the art in the patent cited must suffice. Legal details, of course, must be obtained from the original patent.

In the study of patent literature, when one compares the ardent hopes of the inventors with the results realized from their efforts, it becomes clear that in the development of an art no one man con-

tributes everything. Each subsequent inventor stands on the shoulders of the one who has gone before. It is only through the summation of the contributions of many inventors that the art as a whole prospers.

On October 19, 1887, a British patent was granted to McArthur and Forrest for the use of dilute cyanide solutions for treating gold and silver ores. The rapid extension of this process, in various forms, all over the world, has almost revolutionized the treatment of ores containing precious metals. It is a remarkable fact, however, that long before this patent was granted there was issued, on February 5, 1867, to J. H. Rae, U. S. patent 61866, for an electrocyanide process. Thus a method of electrodeposition from cyanide solutions used for treating ores was invented 20 years before the McArthur and Forrest patent was granted, although the process did not come into general use.

On May 24, 1889, U. S. patent 403202 was granted to McArthur and Forrest for the use of zinc shavings for removing gold and silver from cyanide solutions. The use of finely divided zinc is still the standard method for recovering gold and silver from cyanide solutions. Since this patent was granted, many attempts have been made to replace the precipitation upon the zinc by methods of electrodeposition. It is the purpose of this report to give an outline of some of these methods, as well as to show the relative advantages and disadvantages of the zinc and the electrical methods of recovering gold and silver from cyanide solutions.

PROCESSES OF ELECTRODEPOSITION.

Processes of electrodeposition may be divided into two classes, according as the action takes place, in the presence or in the absence of the pulverized ore itself.

ELECTRODEPOSITION IN THE PRESENCE OF THE ORE PULP.

All the processes of this type hitherto invented require keeping the ore pulp in motion by mechanical agitation. This is necessary, because otherwise the electrical resistance would be too great and because the metal would be deposited in filamentary form between the ore particles and would be difficult to recover.

PATENTED PROCESSES.

The first of these processes was that of J. H. Rae, previously mentioned. The patent discloses the treatment of pulverized ore in a vessel containing the ore together with a solution of cyanide of potassium. In the vessel was provided a metallic agitator which acted as

an anode and kept the ore in suspension. The gold and silver were precipitated on a plate cathode suspended in the solution above the anode. Some time in the seventies a number of experiments were made with such a process in a modified form in San Francisco with ores from Virginia City. The experiments were conducted in an amalgamating pan, the agitator being connected with one pole of the battery and the quicksilver with the other pole. When these experiments were made, it is reported that an alternating current was used, although the inventor expressly states that a direct current should be used. The results obtained were claimed to be slightly better than those with plain amalgamation, but were not enough better to warrant continuing the process.*

On November 3, 1891, a patent (U. S. patent 162535) was granted to the celebrated Prof. William Crookes, of London. This patent was for an electroamalgamation process, by which the ore was to be suspended in a solution containing a soluble salt of mercury, such as sulphate, nitrate, or cyanide. The patent claimed that when suspended ore was submitted to the action of an alternating current of electricity the gold was rapidly amalgamated. I do not know whether this process has ever been tried commercially, but from my own incomplete study of it I do not think that it is likely to be successful. Other and more recent patents are the following:

H. F. Edwards, U. S. patent 518543, April 17, 1894.—An amalgamating device consisting of a vessel containing carbon anodes and mercury cathodes. The solution consists of cyanide of potassium and ferrous oxide. The use of ferrous oxide with cyanide is enough to put this process out of consideration.

Paul Danckwardt, U. S. patent 526099, September 18, 1894.—A process for extracting gold and silver from ores. The appliance consists of concentric drums, either the outer or the inner one of which is made to revolve, either the outer or the inner serving as an anode or a cathode. The suggested use of alkaline sulphide in connection with cyanide of potassium seems entirely out of place; and no better mechanical devices could be found for scouring the precipitated metal from the cathodes.

Pelatan and Clerici were granted three United States patents as follows: No. 551648, December 17, 1895; No. 553816, January 28, 1896; and No. 568099, December 22, 1896. These patents all disclosed cast-iron or steel agitators which acted as anodes, and amalgamated copper plates or a bath of mercury which served as the cathode. They differed from one another merely in the mechanical disposition of the agitator. The first patent showed an agitator of horizontal axis, the second one of vertical axis. The agitator described in the

*Julian, H. F., and Smart, Edgar, Cyaniding of gold and silver ores, 1904, p. 143.

third comprised a belt made of steel with attached arms, so that when the belt was moved by rollers, the stirrers were carried over the surface of the mercury so as to agitate the pulp continuously, the operation being carried on in the presence of a cyanide solution. Experiments on a working scale with this process were conducted by D. B. Huntley in the Delamar mine, in Idaho, and I learn from him that extractions of more than 80 per cent were obtained; but I understand that none of these methods is now in use. The chief objections were the large amount of power used in fine grinding, and the scouring of the cathode. The process was replaced by cheaper leaching methods.

E. Motz, U. S. patent 582077, May 4, 1897.—This covers an electrolytic, rotated drum. The ore and pulp are mixed.

B. Becker, U. S. patent 588740, August 24, 1897.—The appliance consists of a conical tank for agitation from below by a centrifugal pump. A part of the overflow passes into a cylindrical tank having a conical bottom and containing radial or concentric vertical anodes and cathodes. The coarse gold is supposed to be amalgamated upon the sides of the first or agitating tank, and the dissolved gold, electrically precipitated, is recovered in the second tank. Circulation is controlled by suitable valves so that only the finest material passes over to the second tank for electrodeposition.

H. S. Badger, U. S. patent 635544, September 19, 1899.—Electro-amalgamating barrel.

L. E. Porter, U. S. patent 639766, December 26, 1899.—Electrolytic barrel.

C. P. Tatro and George Delius, U. S. patents 640717 and 640718, March 27, 1899.—Apparatus and process.

W. Witter, U. S. patent 641571, January 16, 1900.—Cyanogen halide produced by electrolysis used later with pulp.

N. L. Turner, U. S. patent 653538, July 10, 1900.—Concentric electrodes in tank.

A. J. Irwin, U. S. patent 689674, December 24, 1901.—An endless metallic belt, as anode, supporting the pulp. The cathodes are above.

H. R. Cassel, U. S. patents 694349 and 694350, March 4, 1902.—Process and apparatus. The process comprises electrolysis of the pulp with salt, followed by the use of cyanide, to extract gold and silver.

F. T. Mumford, U. S. patent 706436, August 5, 1902.—The pulp is contained in an amalgamating cylinder of copper-lined steel. The anodes are of carbon.

W. H. Adams, jr., U. S. patent 745742, December 1, 1903.—A tank is provided having a conical bottom and an agitating device. No claims are made for any electrolytic device, but one is disclosed in the

specifications. On account of the lack of proper means for circulating the electrolyte between the inclined electrodes the process would probably be ineffective.

Ernest A. Fahrig, U. S. patent 756223, April 5, 1904.—A complicated system of agitating pulp in a series of inclined electrodes placed in a tower down which the muddy solution falls. The construction is such that an electrically efficient process seems impossible, scouring and other interferences being sure to take place. An apparatus so arranged is not easily constructed or adjusted.

E. L. Oliver, U. S. patent 784120, March 7, 1905.—An apparatus for cyanide-pulp treatment. A cylindrical tank having a conical bottom contains vertical cathodes of copper plate in electric contact with a horizontal tube containing mercury, in such a manner as to allow the mercury to spread by diffusion over the plate as the gold and silver amalgam form upon it. The anodes first specified were $\frac{1}{4}$ -inch sheet-iron plates, but these were afterwards changed to rods of Acheson graphite. The pulp was agitated by compressed air which entered at the center of the conical bottom and forced the ore up through a central pipe, causing it to spread out in the solution and fall between the electrodes. This process was introduced on a working scale at the North Star mine in 1903 and was continued in use there until 1906, during which time more than \$200,000 in bullion was recovered at a profit.* The gold amalgam was recovered in a stronger cyanide solution by changing the cathodes into anodes (after U. S. patent 643096, S. B. Christy). The process gave good results for the treatment of low-grade tailings. An actual recovery of about 75 per cent was claimed. The difficulty in the operation of the process was that the copper plates and the quicksilver held back valuable metal. There was also considerable gold left on the anodes, particularly when the anodes were of iron. Many of the anodes assayed \$70 per ton, and even as high as \$200 per ton. Probably, also, there was a considerable loss of quicksilver. The Acheson-graphite anodes worked very well with tailings; but when an attempt was made to treat sulphide concentrate, sulphates formed and rapidly oxidized the graphite rods which soon were converted into a black mud. As long as sulphates were absent the rods were hardly acted upon. The process of precipitation at the North Star mine has since been replaced by C. H. Merrill's zinc-dust process.

W. A. Hendryx, U. S. patent 785214, March 21, 1905.—An apparatus for the extraction of metals from ores. The apparatus disclosed in the patent shows a vertical cylindrical tank with a conical bottom containing a central tube, in which is placed a revolving screw agitator which discharges the pulp in a continuous stream at the

* Swaren, J. W., Historical notes on the air-lift agitator: Min. and Sci. Press, vol. 108, Sept. 30, 1911, pp. 410, 411.

top over a slightly conical deflecting apron so as to expose the pulp in a thin film to the action of the air. The pulp then descends through a series of electrodes inclined at a slight angle. The anodes are of carbon and the cathodes are of thin sheet lead.

W. A. Hendryx, U. S. patent 836380, November 20, 1906.—An ore-treating process which incidentally required the electrolytic apparatus previously described. The process was introduced in a number of places, but to the best of my knowledge the electrical part has been abandoned and the apparatus is now in use only as an agitator.

J. A. Comer, U. S. patent 833999, October 23, 1906.—A patent applying chiefly to the handling of pulp by compressed air. The electrical part of the device embraces nothing novel.

E. J. Garvin, U. S. patents 809939, January 16, 1906, and 822940, June 12, 1906.*—The apparatus consists of a cylindrical agitator having a conical bottom and containing a small, inverted, floating cone in the center and a small funnel-shaped hopper at the top. The pulp is made to circulate by a centrifugal pump from the bottom of the conical tank over a series of amalgamation plates, electrically connected, which are employed for amalgamating any coarse gold. The pulp then flows through the hopper, down through the agitator, and back through the pump.

The upper part of this apparatus contains nearly clear water that has been allowed to by-pass over the top of the agitator to the electrical precipitation device, which is contained in a rectangular box with a cylindrical bottom. At the bottom of the cylinder is placed a small quantity of quicksilver; and two anodes of sheet iron are connected on either side of the cathode. This cathode consists of a link belt, made of horizontal copper strips, which moves slowly with its lower end submerged in quicksilver, thus keeping the surface amalgamated. The solution passes underneath the first anode, up through the lattice belt cathode, and thence down and out. By means of suitable valves, clear or nearly clear water passes through the precipitation tank. The coarse gold is supposed to be caught on the amalgamated copper plates on the top of the box, and the dissolved gold on the copper-belt cathode of lattice work.

Actual working results by this process are not given in the description, and seemingly it would be difficult and expensive, with this apparatus, to get sufficient area for effective precipitation. Whether it would be possible to prevent mechanical loss, either of the quicksilver or of the gold, from the cathode is perhaps questionable, especially as the descriptive circular states that the liquor flowing into the precipitation box is not always clear, but is usually somewhat turbid. The company at Portland, Oreg., formed for operating the process is said to have abandoned it.

* Garvin Cyanide Extraction Co., Bull. 2, 1906.

DIFFICULTIES OF RECOVERING GOLD AND SILVER FROM CYANIDE SOLUTION
IN PRESENCE OF ORE PULP.

The idea of recovering gold and silver from cyanide solutions in the presence of the ore pulp appears at first sight attractive, as it offers a means of doing away entirely with the great difficulties of filtration and decantation; but all processes founded on this idea have to overcome other obstacles. The first difficulty is that the electrical resistance of a bath containing a large number of non-conducting particles, such as quartz, is greatly increased over that of a clear solution. A second is that the deposited metal is rapidly stripped from the cathodes by the scouring action of the moving ore particles. Such mechanical abrasion causes serious loss. When the cathodes are of mercury or amalgamated copper, a third difficulty results from the surface tension of the mercury.

I contended with this difficulty in experiments made in June, 1889. In these experiments I used as an anode a platinum dish containing roasted pyritic ore that assayed 3.38 ounces gold and 2.63 ounces silver. The dish contained, besides the ore, a 10 per cent solution of cyanide of potassium. The battery used consisted of several dry cells, and the current was 3 milliamperes. The time was 20 hours. I found that a loosely adherent black film formed on the amalgamated copper cathode. The film was easily shaken off into the ore below. The extraction, by difference, was 86.7 per cent. I repeated the same experiment the next day, using as before a platinum dish as an anode, with a 10 per cent solution of cyanide of potassium, and above the ore I placed a small flat porcelain dish (diameter 40 mm.) containing mercury, into which was inserted a graphite rod connected with the negative pole of the battery. The gold and silver and a little copper came down on the mercury in little hair-like particles of about the size and shape of the upper part of an exclamation point (!). They stood all over the surface of the mercury in an erect position like quills on "the fretful porcupine," being buoyed up by bubbles of hydrogen; and owing to the surface tension of the mercury amalgamation of these particles was difficult. Violent agitation would sweep the particles completely away from the mercury surface, whereas gentle agitation would sometimes cause them to amalgamate.

The experiments demonstrated that there are serious difficulties connected with recovering gold and silver from cyanide solutions in the presence of the moving ore pulp. Although a fairly good recovery by the method mentioned is not impossible, a really satisfactory method has not yet been devised.

In confirmation of these views the experiments of Rose* may be quoted. Rose found that with cathodes of mercury or amalgamated copper, and with a current density greater than 0.01 ampere per square foot, a part of the gold came down as a black scum floating on the surface of the cathode in such a form that it could be brushed off with a feather. In the successful experiment with amalgamated copper cathodes there were used 0.7 volt, 0.01 ampere per square foot, 0.082 per cent KCy, and a gold content of 20 ounces per ton. With a lower density the plate remained bright, but the precipitation was extremely slow. In one test, in which mercury alone was used, and with a density ranging up to 0.027 ampere per square foot, less than 1 per cent of the gold appeared in the form of the black scum. With denser currents fully half the gold came down in the form of this scum, which could not be amalgamated except by the use of sodium amalgam. When amalgamated copper was used as the cathode, the copper dissolved in the cyanide, in spite of the current, and caused the destruction of cyanide. Hence, in view of the high cost of mercury and copper, and the difficulty of removing the gold from the copper, the methods requiring mercury or amalgamated copper as cathodes do not seem to have a bright future.

ELECTRODEPOSITION OF GOLD AND SILVER FROM CLEAR CYANIDE SOLUTIONS.

Clear solutions are obtained by either decanting or filtering the ore-bearing solutions, so that only clear solutions containing gold, silver, and possibly other metals, as double cyanides, have to be treated. In spite of the difficulties and expense of decantation and filtration, these steps seem to be necessary for satisfactory results with either electric precipitation or zinc precipitation. However, before going into a critical study of methods, two questions should be answered, namely, What is the valency of gold in cyanide solutions? At what voltage are metals best precipitated from cyanide solutions?

VALENCE OF GOLD IN ELECTROLYSIS OF CYANIDE SOLUTIONS.

It is generally admitted that the gold in cyanide solutions from ore treatment is monovalent, as KAuCy_2 , but there seems to be a general assumption that in electrolysis gold is always trivalent. In Foster's electrical engineer's pocketbook^b the atomic weight of gold is wrongly given as 195.7, and that of silver as 107.11. Gold is plainly stated to be trivalent, and the amount precipitated per ampere-hour is given as 2.4512 grams, and the amount of silver precipitated per ampere-hour is given as 4.0248 grams.

* Rose, T. K., The electrical precipitation of gold on amalgamated copper plates: Trans. Inst. Min. and Met., London, vol. 8, 1900, p. 369.

^b Compiled by R. S. Woodward, Jr., and G. S. Miller, Jr., 1910, p. 1230.

Roeber* does not commit himself on the valence of gold, but gives the reader a free choice among three values. Roeber's article is one of the best short statements of modern electrochemical theories. His data in regard to silver and gold are as follows:

Atomic weight of silver, 107.93; valence 1, ampere-hour 4.025 grams.

Atomic weight of gold, 197.2; valence 3, ampere-hour 2.451 grams; valence 2, ampere-hour 3.677 grams; valence 1, ampere-hour 7.35 grams.

"Hütte." Taschenbuch für Eisenhüttenleute (1902 edition) gives for silver: Atomic weight, 107.7; valence 1, ampere-hour 4.0269 grams. The values for gold are: Atomic weight, 196.2; valence 3, ampere-hour, 2.4453 grams.

Steinach and Buchner^b give the following values: Silver, monovalent, ampere-hour 4.026 grams; gold, trivalent, ampere-hour 2.445 grams.

Keith^c says that gold is trivalent in cyanide solutions.

According to the International Atomic Weights Committee^d the atomic weight of silver is 107.93, with a possible error not greater than ± 0.01 , and the atomic weight of gold is 197.2, with a possible error not greater than ± 0.1 . These values being assumed to be correct, if gold is monovalent, for every grain of silver precipitated by a given current 1.837 grams of gold should be precipitated by the same current, and if gold is trivalent, for each gram of silver precipitated there should be precipitated only 0.612 gram of gold.

Of course gold is capable of reacting in at least two cyanide compounds— KAuCy_2 and KAuCy_4 —the former of which is monovalent and the latter trivalent. In estimating the electrical efficiency of cyanide precipitations, it is absolutely necessary to know which of these values to choose for the reactions that take place. To investigate this matter I prepared three separate solutions, as follows: No. 1 containing 250 mg. of silver as nitrate, No. 2 containing 250 mg. of silver as KAgCy_2 and 150 mg. of free KCy , and No. 3 containing 100 mg. of gold as KAuCy_2 and 150 mg. of free KCy . The anodes and cathodes in solutions 1 and 2 were both of pure silver. In solution 3 they were of pure gold. Three cells were thus set up in series with a storage-battery cell of 2 volts, and after a period of 1 hour and 50 minutes all the electrodes were weighed. The following results were obtained:

*Roeber, E. F., Standard handbook for electrical engineers, 1908, 1,285 pp.

^bSteinach, Hubert, and Buchner, Georg, Die galvanischen Metallniederschläge, und deren Ausführung, Berlin, 1890, 258 pp.

^cKeith, N. S., Electrolysis of gold: Jour. Inst. Elec. Eng., vol. 24, 1895, pp. 236-260.

^dClarke, F. W., Seubert, K. E., and Thorpe, J. E., Report of the International Committee on Atomic Weights, 1903, p. 5.

Experiment 1. The anode (A) lost 45.23 mg., and the cathode (B) gained 43.22 mg. The silver came down in fine crystals, and a small proportion of these was unavoidably lost in weighing, which accounts for the low value obtained.

Experiment 2. The silver anode (C) lost 45.12 mg., and the cathode (D) gained 43.78 mg.

Experiment 3. The gold anode (E) lost 81.52 mg., and the cathode (F) gained 76.69 mg. It will be noticed that the gain in each test was slightly less than the loss, even in experiment 1, in which there was mechanical loss in weighing the cathodes.

Let us compare the gain of the gold cathode F with that of the silver-nitrate cathode B. Dividing F by B, we have 1.774, and comparing the gain of the gold cathode F with that of the silver cathode D, both values being in cyanide of silver, we have $\frac{F}{D} = 1.75$.

However, as there might be a slight loss in weighing the cathodes, it might be nearer the truth to compare the losses of the anodes rather than the gains of the cathodes. First let us compare the gold loss from anode E with the silver loss from anode C, both in cyanide solutions. If we divide E by C the quotient will be 1.807 instead of 1.837. Supposing the metal to be monovalent, if we divide E by A we have 1.802 instead of 1.837.

It must be evident therefore that in aurocyanide of potassium solutions the gold acts practically as a monovalent metal. That less gold is precipitated than is dissolved in cyanide solutions is probably due to the presence of free cyanide of potassium. It is necessary however to have a certain excess of cyanide, as if there is no excess, AuCy_2 forms on the anodes.

Iron is sometimes trivalent and sometimes bivalent, and it is possible that similarly gold may be both monovalent and trivalent. It is well known that gold does exist as potassium auricyanide, but it appears that the aurocyanide salt is much more stable, the auricyanide tending to break down into the aurocyanide. However, there seems to be no reasonable doubt that in estimating the electrical efficiency in precipitation processes the gold should be considered as monovalent.

VOLTAGES AT WHICH THE METAL IS COMPLETELY PRECIPITATED FROM CYANIDES.

This is a subject about which little has been published. Friedenberg* claims that the precipitation is as follows:

K_2HgCy_2 , complete at 1.6 volts.

KAgCy_2 , complete at 1.7 to 1.8 volts.

* Friedenberg, Ztschr. Phys. Chem., Bd. 12, 1891, p. 97.

KAuCy , complete at 1.9 volta.

K_2CuCy_4 (with $\frac{1}{2}$ to 1 part KCy) at 2 volta.

KCuCy , (with 10 to 15 parts KCy) at 2.5 volta.

Some precipitation is possible at lower voltages than these. Theoretically, complete precipitation would require an infinite voltage, but what is meant by the author quoted is precipitation sufficiently complete for quantitative purposes. From my experience I consider the voltage given for copper entirely too low. Experience shows that much depends on the amount of free cyanide and other salts present.

The figures cited are to be regarded only as lower limits. The author states that the precipitation is slow at these voltages and prefers to use 2.5 volts for gold. The figures explain the low results obtained in the experiments with the Keith process in which an average voltage of only 0.75 was used.

SIEMENS & HALSKE PROCESS.

In 1888 Wernher von Siemens first applied for patents (Heirs of E. W. von Siemens, U. S. patent 601068, Mar. 22, 1898) for his process of electrical precipitation which seems not to have been introduced on a working scale until 1893. The process had previously been patented in the Transvaal under date of July 7, 1892, patent 397. The patent covers both the method and the apparatus, according to the patent laws then in force. The precipitation method claimed is as follows: "A method of extracting gold from a weak cyanide solution, which consists in circulating the solution over anodes of iron and cathodes of lead. Said cathodes being formed of thin plates arranged at short distances apart, and have from 9 to 10 square feet of surface for each ton of solution in contact with them. Said solution while in motion is subjected to an electrical current of from $3\frac{1}{2}$ to 4 volts and 0.5 to 1.5 amperes per square meter of cathode surface substantially as set forth." It will be observed that the claims allowed are not general but are much restricted in their nature.

The Siemens process was first introduced in South Africa through the agency of Charles Butters, and although the patent shows a centrifugal pump as the means of continuously circulating the solution through the deposition box, this was soon abandoned and the process was applied by constructing long boxes to allow the gold and silver to be deposited sufficiently by a single flow through the box. The anodes were of iron, and were placed 3 inches from the cathodes, which were made of thin strips of lead. As soon as 2 to 12 per cent of the gold, or on an average 7 to 8 per cent, had been deposited on

the lead cathodes, the lead strips were taken out from the cells, melted down, and cupelled.

This process, popularly known as the "Siemens & Halske process," was used for a long time in the Transvaal in competition with precipitation by zinc shavings, and was fairly successful there under the management of Mr. Butters. However, after he left the Transvaal, the method was gradually replaced by precipitation by zinc shavings, the latter method having been greatly improved. For some time the Siemens & Halske process has been substantially abandoned in Africa. In a modified form, mentioned later, electrical precipitation is still used by Mr. Butters in several of the mines under his management.

One difficulty with the Siemens & Halske process was the large size of the box necessary for effective precipitation. Mr. Butters told me that when he first constructed the cells some of his English friends asked, "Are you constructing stalls for horses?" The cells were large enough to have been used for that purpose. One of the reasons for the large size was the necessity of avoiding short circuits. These were liable to be encountered owing to the fact that the anodes, in the presence of sulphates and chlorides, contained in the solution, soon became coated with warty growths of hydrate and hydrated carbonate of iron and prussian blue. Hence a 3-inch space was left between the anodes and the cathodes. The maximum daily capacity was not usually more than 5 tons of solution treated to each box holding 1 ton of solution, and the volume treated was usually much less.

There have been many descriptions of the Siemens & Halske process, but most of them omit some detail necessary for calculation of the electrical efficiency. The most complete description of the essential details of the process that I have been able to find is that by Richards.*

There were four deposition boxes, each 30 feet long by 4 feet 9 inches wide and 3 feet deep. Each box contained 110 iron anodes and 110 cathodes formed of bags containing 5 to 10 pounds of lead shavings the extent of the surface of which is problematical. The tanks were electrically coupled in series, 2.5 volts being required for each. The current was 200 amperes. Each box held about 14 tons of solution, or, allowing for the electrodes, 10 to 12 tons, so that the capacity of a box holding 1 ton of solution would have been 4 to 5 tons a day. The heads assayed 80 and the tails 15 grains gold. Hence, only about 81.5 per cent of the gold content, weighing 3,250 grains (0.464 pound or 0.21 kilogram), was precipitated at this rate of flow per day. Prof. Richards comments as follows:

That this is an insignificant amount for the current passing (200 amperes), which should theoretically deposit 34.56 kilograms in each box, is patent to the

* Richards, J. W., The cyanide process for the treatment of gold ores: *Jour. Franklin Inst.*, vol. 143, February, 1897, p. 96.

electrometallurgist, the yield being only 0.6 per cent, but it must be remembered that the extraordinarily dilute solution used will not yield up its gold any faster and that it must be electrolyzed slowly to get any sort of correspondence between the power used and the gold obtained.

Under the conditions of the Siemens & Halske process this statement is true, but, as shown later, under the right conditions better results can be obtained.

If we assume as correct that 7.35 grams of gold should be precipitated in 1 ampere-hour, it follows that 200 amperes should precipitate 1,470 grams each hour, or in 24 hours there should have been precipitated 35.28 kg. instead of 0.21, showing that the efficiency was only 0.59 per cent, which is essentially the same as that of Prof. Richards, who seemingly used a different constant. The gold recovery also is rather low, there being a reduction only from \$4.26 to \$0.64 per ton.

Julian and Smart's excellent book,* giving details of the Siemens & Halske method of electrical precipitation, has been carefully examined to see whether it presented data that could be used in calculating the electrical efficiency of the process. Although Table 31 contains 18 columns of data, it is not complete enough for a reliable calculation of the electrical efficiency.

However, by making some assumptions, approximations may be possible.

Thus a high precipitation is cited as being obtained at the Croesus works where seemingly the gold content is said to be reduced from 48 grains to a trace. However, the authors do not state the current density per square foot of cathode surface, but it probably was not higher than that cited for other operations, namely, 0.04 ampere per square foot. Only one precipitation box, containing 38.8 tons of solution, was used, 50 tons being treated in 24 hours. On the basis of a 1-ton box the rate would be 1.29 tons of solution per 24 hours. The cathode area is given as 7,680 square feet. If we assume the current density to be 0.04 ampere per square foot we shall have 307 amperes, or 7,368 ampere-hours, a day. Precipitation should then be at the rate of 7.35 grams of gold per ampere-hour, or 54.155 kg. of gold. The heads are said to have contained 48 grains and the tails a trace. If we assume all of the gold, 48 grains per ton, to have been precipitated, the recovery for the 50 tons of solution would be 2,400 grains, or 0.156 kg. of gold actually precipitated. Dividing 0.156 by 54.155 gives only 0.288 per cent as an electrical efficiency. Hence, although the recovery is rather high, the capacity per ton of box capacity is only 1.3 tons of solution a day, and the electrical efficiency is less than 0.3 per cent, which is one-half that calculated by Richards.

* Julian, H. F., and Smart, Edgar, Cyaniding gold and silver ores, 1904, p. 143.

The figures as presented by Julian and Smart for the May Consolidated plant may next be considered. Two results are given—the one covering the solution coming from the sand tank, and the other covering the solution from the slimes. In the sand tank 6 boxes were used, containing 133.2 tons of solution. These treat 300 tons a day. Consequently the capacity on the basis of a 1-ton box is 2.25 tons a day. The cathode surface is not given. If we assume it to be the same as the anode surface, 13,824 square feet, with a current density of 0.04 ampere, we shall have a total of 553 amperes, or in 24 hours, 13,271 ampere-hours. This current should precipitate 97,542 kg. of gold. As a matter of fact the head solution assayed 120 grains of gold per ton and the tail solution 18 grains, or there was precipitated 102 grains per ton. For the 300 tons, with the weights converted to kilograms, there was precipitated 1,982 kg. of gold, or the electrical efficiency was 2.03 per cent, according to the data given and the assumptions here made. But it is altogether probable that the cathode surface was much larger than that of the anodes. Hence the efficiency would be much less than that stated. The precipitation is rather high, being over 93 per cent, but the capacity is low, being one-half of what I obtained with my experiments at Bodie.

Considering the May Consolidated slime plant, we find that the solution was treated in four boxes, containing 181 tons, in which 320 tons was treated each day. At this rate a 1-ton box would have a capacity in 24 hours of 1.73 tons, a lower rate than was obtained from the sand solution. There was 16,892 square feet of anode surface. The cathode surface is not given, but was probably larger than that of the anode surface, and if the surface of each be assumed to have been the same, with a current density of 0.04 ampere, the daily current of 16,220 ampere-hours thus obtained should have precipitated 119.22 kg. of gold. As a matter of fact the head solution contained only 24 grains of gold and the tail solution 2.4 grains. There was actually recovered, per ton, 21.6 grains of gold, or, from 320 tons there was precipitated 0.448 kg. of gold. This when divided by 119.2 gives only 0.376 per cent electrical efficiency. If, as is probable, the cathode surface was larger than the anode surface, the electrical efficiency would be still lower.

MOLLOY PROCESS.

The following description of the Molloy process is taken from Julian and Smart:^a

This process was patented in the Transvaal in 1892 by B. C. Molloy, where it was employed for some months. It consists in passing the gold-bearing solution over a surface of mercury in which sodium was continuously deposited.

^a Julian, H. F., and Smart, Edgar, Cyaniding gold and silver ores, 1904, p. 158.

An ingeniously devised apparatus was employed which consisted of a large shallow tray covered with mercury, down the center of which was a narrow compartment, with sides that dipped slightly into the mercury. In this compartment was placed a strong solution of sodium carbonate which was electrolyzed, the mercury at the bottom being the cathode, and the anode was of peroxidized lead. As the sodium deposited and accumulated in the mercury within the compartment it diffused to the mercury on the outside, over which a constant stream of gold-bearing solution was kept flowing. The gold-bearing molecules dissolved as they came in contact with the sodium amalgam surface, were decomposed, and gold amalgam formed.

The process was ably developed by Dr. A. Simon, who employed it on a working scale in the treatment of tailings on Witwatersrand in 1892-93.

The weak point of the the process lies in the difficulty of getting a sufficiently large electrode surface for the solution to come in contact with, without unduly increasing the amount of mercury handled. Thus, if the area of a surface required to reduce a 5 dw't. solution down to 1 dw't. was 1, that to reduce a 1 dw't. solution down to 5 gra. would have to be somewhere about 25 with the same rate of flow, showing how enormously the area has to be increased as the solution becomes impoverished. This difficulty may, however, be got over, to some extent, by agitating the solution rapidly over the sodium amalgam surface as it flows, by mechanical means. The efficiency of sodium amalgam as a precipitant is small, but when employed in a dilute state, as in the Molloy process, it compares favorably with the electrical precipitation processes of to-day. Sodium amalgam is said to act on KCy in solution forming complexes, but in dilute working solution this action is inappreciable.

I have obtained good precipitation by this method so long as the capacity was limited, but the moment it was pushed, the extraction fell off. Of course, all the objections to the use of mercury apply. The process seems to have been abandoned.

VARIOUS OTHER PATENTED PROCESSES.

Johannes Pflieger, British patent 16737, September 6, 1895.—The inventor uses cathodes composed of shavings, wire, powder, or shot and presenting a large surface. He prefers iron-wire cloth of $\frac{1}{8}$ -inch to $\frac{1}{2}$ -inch mesh, but finds that 10 to 30 wires per linear inch answer the requirements. The specifications show a deposition box containing an interior compartment filled with wire cloth, and provided with transverse partitions so arranged as to force the solution alternately down and up through the wire cloth. On either side of this interior compartment are two outer compartments separated from the inner one by diaphragms. Outside the diaphragms the vessel contains a 1 to 5 per cent solution of soda, and in this latter solution zinc-plate anodes are placed, forming with the cathodes a galvanic cell. This cell, when the anodes and the cathodes are connected, forms a circuit which, it is claimed, will rapidly cause the metal to be deposited on the wire cloth, and the zinc plates to dissolve in the soda solution. It is evident that in this apparatus no exterior electrical power is used. The electromotive force available is limited to 0.6 to 1 volt.

Julian and Smart^a give the capacity of a 1-ton box as 16 tons per 24 hours. The process seems to have been abandoned.

Oscar Froelich, U. S. Patent 556092, March 10, 1896.—Lead cathodes and carbon anodes are used. The current density is 6 amperes per square yard. The solution is seemingly clear.

Rudolf Keck, U. S. Patent 566986, September 1, 1896.—Clear solution, lead anodes, and aluminum cathodes.

Emile¹ Andreoli, U. S. Patent 568724, October 6, 1896.—An apparatus for the electrical deposition of gold and silver in a tank provided with one or more anodes and a series of amalgamated cathodes. Each cathode consists of a pervious skeleton of knit-wire plates upon a layer of mercury in the bottom of the tank, into which the cathode dips. This patent discloses the use of peroxidized lead as anodes. The solution used is clear. Anodes are contained in compartments separated from the cathodes by permeable membranes. According to the description, only two anodes are used. The inventor places the anodes on the sides of the tank. The cathodes dip into a bath of mercury at the bottom. The use of the network of wire gauze is specified. The diaphragms are made preferably of porous asbestos or kieselguhr (infusorial earth) porcelain, but woven tissues stretched on frames may be used. It is specifically stated that the porous partitions may be omitted, and that the cathodes may be placed crosswise, one perforated anode being used for 12 to 15 cathodes. The patent shows a great advance in the art, particularly the use of peroxidized-lead anodes, which is a valuable discovery. The use of the wire cloth is also good, but the process necessitates the use of a large volume of quicksilver in the bath, which is a serious disadvantage, although it is convenient in making the electrical connections. However, the process does not appear to be practicable on account of the great difficulty of removing the gold and silver from the cathodes.

Andreoli^b has written an interesting article on the electrical deposition and recovery of gold. In this article he explains his discovery of the peroxidized-lead anode. He states that this anode was not the result of research or study, but was simply a chance discovery. His first plates were made according to the Plante method, but, at present, he prefers to use plumbate of soda repeatedly to coat the plates with peroxidized lead. When so treated the plates are drawn, washed, and placed in a strong solution of cyanide of potassium, where, under a heavy current, they become hard and acquire a crystalline appearance. In reference to his process Andreoli states: "I was soon disappointed when I was told that no one in mining districts would care much to adopt such a process which involved too much

^a Julian, H. F., and Smart, Edgar, Cyaniding gold and silver ores, 1904, p. 120.

^b Andreoli, E., Electrodeposition and recovery of gold: Jour. Soc. Chem. Ind., vol. 16, Feb. 27, 1897, pp. 96-97.

trouble in distilling the mercury and cleaning the plates in order to recover the gold amalgamated with it." He therefore altered the process by using cathodes of iron plates, upon which the gold and silver were deposited. He then recovered the gold and silver from the iron plates by dipping them in molten lead, whereupon the gold and silver alloyed with the lead and the plates were stripped. However, the iron plates are said to have become warped by the treatment with the hot lead, so that they easily produced short circuits. The only part of the Andreoli process that has been permanently adopted is the use of the peroxidized-lead anodes.

Max Netto, U. S. Patent 573233, December 15, 1896—This process consists in precipitating silver and gold from alkali-cyanide solutions by acidulating with hydrochloric acid so as to precipitate silver chloride, separating the precipitated silver chloride by filtration from the solution, and subjecting the acid filtrate to the action of an electric current so as to deposit the gold on the cathode. The inventor regenerates the cyanide solution by the addition of caustic alkali.

Precipitation of silver and gold in the presence of cyanide of potassium by hydrochloric acid is seldom complete. If the solution is high in cyanide and low in gold no precipitation of gold occurs; but if the solution contains a small amount of cyanide and a large amount of gold and silver, a certain amount of gold always comes down with the silver.* On account of these reasons and the cost of the acid and alkali, it would seem that the process is hardly practicable, except where acid and alkali are much cheaper than cyanide.

J. F. Webb, U. S. Patent 585492, June 29, 1897—A method and apparatus for separating precious metals from their solvent solutions. The specifications show a box containing alternate vertical columns of zinc shavings and carbon. The carbon is said to be either charcoal or coke. The solution passes down through the charcoal and up through the zinc, the first compartment containing carbon and the last one zinc. The terminal columns of carbon and zinc are connected by wire, and electrical action between the carbon and the zinc is claimed. The inventor does not, however, rely upon this connection alone, claiming that the zinc and charcoal act sufficiently without the connection. As ordinary charcoal is not a conductor of electricity, there can be here electrical action only between the coke and the zinc, and the use of an external source of current for the process can not be claimed. Practically only local action can take place.

E. Andreoli, U. S. Patent 598193, February 1, 1898—An apparatus for the electrical deposition of gold, silver, and other precious

* See Christy, S. B., Solution and precipitation of cyanide of gold: *Trans., Am. Inst. Mining, Eng.*, vol. 26, 1896, pp. 747-748.

metals. The inventor uses anodes of peroxidized lead acting in the presence of and in combination with the cyanide solution. The cathodes are made of perforated zinc plates.

The first great improvement over the Siemens & Halske process was the introduction of the peroxidized lead anodes discovered by Andreoli. When the lead plate was covered with a perfectly continuous film of peroxide of lead the anodes were practically unattacked by the cyanide solution, but whenever this film became broken, so as to expose the metallic lead, a white funguslike growth of hydrate and cyanide of lead formed upon the lead which was almost as destructive as the hydrate of iron, that forms on the iron plate. When this action takes place, lead anodes are soon corroded and finally fall to pieces. Mercury also rots lead plates and soon destroys them. Experiments that I have made show that, in cyanide of potassium, the peroxide-of-lead anodes are practically unacted on when new, but that, in the presence of potassic hydrate, they dissolve slightly and a trace of lead comes down on the cathode.

The solubility of PbO_2 in KC_y and KHO was shown in experiments made November 26, 1900. I took as an anode an old peroxidized plate from a storage battery and as a solution 0.2 per cent potassium cyanide. As a cathode I used a platinum sheet $6\frac{1}{2}$ by 8 centimeters. The weight of the platinum cathode before treatment was 18.6734 grams. When treated with a voltage of 4 volts and a current of 0.068 to 0.11 ampere for two hours the cathode of platinum, which had previously been carefully cleaned, actually lost 2 mg. in weight. This result was typical of many experiments, and the results of other experimenters show similarly a slight loss of platinum from cathodes in cyanide electrolysis. Certainly at this time no lead was deposited. To the solution was then added KHO to bring the total to 0.1 per cent free KHO . For two hours the voltage was again 4, and the current 0.171 to 0.070 ampere. The gain in weight of the platinum cathode was 0.7 mg., caused probably by deposited lead. I then added salt to make 0.2 per cent NaCl and continued the treatment for 16 hours with a voltage of 3.7 to 3.42 volts, and a current at 0.2 to 0.56 ampere. The gain in weight of the cathode was 8.2 mg. The conclusion was that dilute solutions of KC_y do not alone attack peroxide of lead in two hours, but that KC_y containing KHO and NaCl do attack it slightly.

Kern* claims that peroxidized lead anodes fail, owing to the electrolyte attacking the lead through the pores in the peroxide coating, but I believe this to be more likely due to cracks in the brittle coating of peroxide caused by the easy bending of the lead within.

* Kern, E. F., Electrolysis of cyanide solution: *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, pp. 241-270.

N. S. Keith, U. S. patent 597820, July 25, 1898—A process for treating ore with a cyanide solution containing cyanide of mercury. The gold and silver are recovered from the filtered solution by passing a current of electricity of less than 1 volt through suitable anodes and cathodes upon which the metal deposits as a coating of amalgam. I saw this process in operation in the year 1898. The ores came from Cripple Creek, Colo., and were treated at Cyanide, Colo. The anodes and cathodes were both of sheet iron, and the solution passed alternately up and down between the electrodes.

EXPERIMENTS WITH KEITH PROCESS AT CYANIDE, COLO.

I am indebted to Mr. Philip Argall for the following data on the use of the Keith process. From March 31 to April 9, 15 tons of solution was used to treat 83 tons of ore, the precipitated solution being used over again. The precipitation process is of chief interest.

The precipitation was conducted in a tank 12.5 by 3 by 4 feet, having a total content of 150 cubic feet, or containing about 5 tons of solution. It treated 7 to 13 tons a day, or an average of 10 tons. That is, a 1-ton tank would have had a capacity of about 2 tons in 24 hours. The anodes and cathodes were of sheet iron 4 by 3 feet. There were 50 cathodes so that the total area on both sides was 1200 square feet. As there were 20 amperes flowing through the box, the current density was about 0.017 ampere per square foot. The potential varied from 0.5 to 0.9, averaging 0.75 volt.

From March 31 to April 9, the solution treated was used in leaching ore. During this time the solution entering the box ran from 1.605 to 0.215 ounces of gold, and that leaving the box ran from 0.385 to 0.107 ounce. On two days it left the box richer than it entered, but as no record is given of voltage and amperes for those days there may have been a defect in the current. The best precipitation was 76 per cent (the first day); the precipitation varied from that figure to 15 per cent and less, or even to negative value.

From April 10 to 21 the solution was no longer used to treat ore, but was pumped back to a tank and allowed to flow repeatedly through the box at the rate of 10 tons per day to indicate what the precipitation process could do. In 12 days gold content of the heads was reduced from 0.135 to 0.050 ounce. The solution was treated at the rate of 2 tons a day in a 1-ton box, or of 12 to 22 pounds per 24 hours per square foot of cathode surface. As the box was 12.5 feet long and there were 51 anodes and 50 cathodes, the average distance between the electrodes was 1.5 inches. The same solution when afterwards passed only once through boxes of zinc shavings was reduced to 0.015 ounce, or about 30 cents a ton. The original solution used by Prof. Keith contained 0.19 per cent KCy and 0.66 per cent NaHO,

and the final solution contained 0.04 per cent KCy and 0.155 per cent NaHO. There was used also 12 pounds of HgO. Mr. Argall comments as follows:

During deposition a black deposit collected on the cathodes, interfering with precipitation. This deposit had to be cleaned off three times. The material collected was chiefly of a graphitic nature, oxide of mercury and gold. The gold recovered from the black mud amounted to 25.7 per cent of the total gold recovered.

The mercury appeared to be the first metal deposited; consequently, the plates from the center to the end of vat were very hard, and the amalgam was extremely difficult to remove. When I decided to make a complete clean-up and abandon the process, some 5 pounds of mercury oxide was dissolved in the solution to soften the plates, and even with this addition of mercury it took two men one week to clean 50 plates, and when they had scoured off all the deposit possible, using mercury freely in rubbing the plates, there still remained on or in the plates 41.5 per cent of the total gold recovered.*

Recapitulation:

	Per cent.
Gold recovered as amalgam.....	32.8
Gold recovered as mud.....	25.7
Plates in 1 per cent cyanide solution.....	41.5

The cathodes were attacked and pitted somewhat. The anodes did not show much action, though a little Prussian blue appeared in places.

The result of this test led to the abandonment of the Keith process. Seemingly, the small amount of action on the anodes, which were nearly covered with a thin, hard scale of ferric oxide, was due to the low voltage used. This was always less than 1 volt. Curiously enough, the best precipitation was from a rich solution, on the first day, when the voltage recorded was only 0.5 volt. According to Mr. Argall the mercuric cyanide seemed slightly to hasten the solution of the gold, but not sufficiently to justify its use. This has been denied. Mr. Claude Vautin states that experiments in Hungary (1893-95) showed that the use of mercuric cyanide actually retarded the solution of the gold. I think that it probably retarded the precipitation of the gold.

In the light of further experience there were shown to be three serious defects in the process as conducted. First, the voltage was too low. The good results obtained upon the first day, with 0.5 volt, were doubtless due to the fact that the cathodes were of naked iron; but as soon as the cathodes became coated with gold, a back electromotive force operated to oppose further deposition of gold. The second defect was the use of mercury, the surface tension of which increased the difficulty of precipitation. The third defect was the sluggish circulation. The difficulties of the clean-up would largely disappear in regular work, just as they do in the use of amalgamated copper plates. It would be as difficult finally to clean them.

* This was subsequently recovered by soaking the plates in a 1 per cent KCy solution.—Author.

FIRST DEVELOPMENT OF THE CHRISTY PROCESS.

When I first looked into the electrodeposition of gold and silver from cyanide solutions it seemed to me that the low ampere-hour efficiency in the Siemens process was a fatal bar to the success of electrical deposition, and I was not surprised that the process was abandoned in South Africa shortly after Mr. Butters left there. But in discussing the matter with the late Stephen D. Field, the well known electrical engineer, my attention was called to the fact that the ordinary telephone has less than 1 per cent of electrical efficiency, and yet it is one of the most successful of modern inventions. This suggestion caused me to attack the problem with renewed interest, as I thought the method might still be made effective if given more careful study. Siemens was well aware of the importance of having a large electrode area, and he especially calls attention to the importance of increasing this rather than the voltage of the bath.

I soon came to the conclusion that all electrical processes that used quicksilver, either in the liquid form or in the form of amalgamated plates, except for treating free gold, coarse enough to overcome the surface tension of the mercury, were not likely to be successful with cyanide solutions. Such processes involve the use of a large amount of expensive metal, difficult to handle without loss, and involve the danger of salivation of the workmen. Moreover, there is the difficulty, already mentioned, arising from the necessity of overcoming the surface tension of the metal. This difficulty was particularly evident in the test of the Keith process at Cyanide, Colo. There was also great difficulty in recovering the amalgam from the large area of the cathode plates necessary for a complete precipitation. I noticed, moreover, that so long as the solution contained no sulphates or chlorides the anode plates, of iron, were little acted upon, being simply covered with a thin film of ferric oxide which protected the metal from further action, although it slightly increased the resistance.

Starting with the principle that an enormous cathode surface was necessary for the rapid and complete precipitation of the gold and silver, I made a number of experiments, using sheet iron for both cathodes and anodes. I found that when the current was not too great the precipitated gold and silver adhered firmly to the surface of the sheet-iron cathode. I then removed the cathodes, two at a time, and made them anodes in a dilute cyanide solution of about 0.1 per cent, and redissolved and reprecipitated the gold electrolytically upon a piece of sheet iron covered with a thin film of graphite and vaseline. On graphiting the surface of the sheet iron I found it possible, under proper conditions, to reprecipitate the gold and

silver in a dense coherent sheet, which afterwards could be stripped in sheets ready for the mint.

On February 6, 1900, I was granted U. S. patent 643096, for "a process of progressive electroconcentration and recovery of gold and silver contained in the large volumes of dilute cyanide solution containing free alkali resulting from the extraction of gold and silver ores, tailings, and concentrates, which consists first in depositing the gold and silver electrolytically from said solution upon removable cathodes sufficiently numerous and large in area to secure efficient deposition, and second, in making said removable original cathodes successively anodes in a smaller volume of cyanide solution, and transferring and depositing electrolytically the thin film of gold and silver already distributed over a large number of said original cathodes upon a smaller number of secondary cathodes, also contained in said smaller volume of cyanide solution."

EXPERIMENTS AT STANDARD CONSOLIDATED MINE, BODIE, CAL.

In the summer of 1900 I tried this process on a small scale at Bodie, Cal. The solution treated was 0.937 short ton, having an assay value of 0.175 ounce of gold and 0.437 ounce of silver. Although the metal values were almost entirely in the gold, there was nearly three times as much silver as gold in the solution. The assay value of the head solution was about \$4 per ton, gold and silver. The average of the entire solution after treatment was only about 10 cents a ton, some assays being as low as 2 cents. All the assays were made by the company assayer, W. M. Knox. Samples of eight assay tons of solution were assayed by evaporation with litharge.

The deposition box used was 2 feet 6 inches by 4 inches by 6 inches, and held 0.4 cubic foot. It treated solution at the rate of 2 cubic feet per day, or five times its capacity, with good results, but on pushing the work beyond the capacity of the box, the value of the tailing would run up to 20 or 30 cents.

There were 27 cathodes of thin sheet iron, 4 by 5 inches; hence their exposed surface on both sides was 7.5 square feet. The current varied from 0.5 to 1.3 amperes, the cathode current density varying between 0.066 and 0.173 ampere per square foot, averaging 0.135 ampere. For a part of the time, two boxes were used, in series, as far as solution was concerned, but electrically in parallel; hence the current density was then only 0.068 ampere. The voltage varied from 0.8 to 3.6 volts. The electrodes were all placed transversely across the box and $\frac{1}{2}$ inch apart, so that the solution flowed alternately up and down the box. Alternate anodes projected above the surface of the solution and were perforated with $\frac{1}{4}$ -inch holes to allow the solution to flow through. The flow through these holes

was more rapid than usual, and I found the deposit opposite these holes to be thicker and better than elsewhere. This discovery first suggested to me the advantage of rapid circulation through pervious electrodes, an idea that was developed later.

In order to preserve the anodes from corrosion they were coated with a mixture first suggested by Mr. Bettal of South Africa. This mixture consists of graphite, litharge, boiled oil, and turpentine; it is put on hot. I found that with a low current density (voltages up to 0.14 volt) the anode coating stood up well, as was claimed for it, but that as soon as the pressure was raised to 2 or 3 volts, in order to increase the precipitation, the boiled oil and turpentine were attacked, producing a vile-smelling mixture. The iron was then so attacked that the anodes had to be scraped clean.

There was considerable trouble in getting a continuous, direct-current service of a proper voltage. Alternating current was brought from a distance and was erratic, so that, although for a part of the time, direct dynamo service was available, frequent resort was necessary to a couple of large improvised Daniell cells.

THE CLEAN-UP.

The clean-up was effected in a cell that held 750 c. c. of cyanide solution, and accommodated two anodes (two cathodes from the deposition box, to be stripped) and one sheet-iron cathode, on which the gold and silver were to be precipitated. This cathode was coated with a layer of graphite and wax, put on hot, and rubbed dry with graphite, to favor the subsequent stripping of the concentrated metal. With two Daniell cells giving 1.4 volts in closed circuit, 200 ampere was obtained, but this fell rapidly to 0.050 ampere. The iron anodes did not appear to corrode, yet some of the metal came down rather loosely on the cathode which became coated evenly with silver and gold. The clean-up solution, which at first was 0.1 per cent KCy, was afterwards made 1 per cent. The cathodes from the head of the deposition box required about 1 hour for stripping, and those from the tail required only a few minutes. The cathodes from the deposition box (anodes in the clean-up box) showed no signs of pitting except in one instance when the current was left on over night. If the stripping was stopped when nearly complete the oxidation of the anodes in the clean-up box could be easily avoided, especially as clean cyanide solution free from chlorides and sulphates could be used for stripping. While the metal was partly adherent to the the cathode, much of it came down as a dark, blackish-brown powder, easy to filter and to melt. In later experiments the deposition of this precipitate was avoided by proper circulation. The scrapings from the anodes of the deposition box after treating

the mill solution contained graphite, linseed oil, etc., with more or less hydrate of iron and Prussian blue; weighed about 316 grams; and carried 0.16262 gram in gold and 0.30785 gram in silver. The total clean-up resulted as follows:

Clean-up in experiments at Bodie, Cal.

	Gold, grams.	Silver, grams.
From solid metal from cathode of clean-up box	1. 61678	4. 14322
From metallic dust from clean-up box	1. 95100	5. 53900
From cyanide solution from clean-up box	. 44346	. 85814
From wash water from anodes in clean-up box	. 14936	. 36704
From various sediments from deposition box	. 16362	. 30785
	4. 32322	11. 21525

The original solution treated was 0.937 ton, which assayed 0.164 troy ounce of gold and 0.435 troy ounce of silver; hence in grams the result was as follows:

	Gold, grams.	Silver, grams.
Original content	5. 11	12. 70
Actual recovery	4. 32	11. 22
Lost in tailing solution, assay samples, etc	. 79	1. 48

An 8-assay-ton charge of the tailing solution from the whole charge gave 0.005 ounce of gold per ton and 0.005 ounce of silver; that is, the tailings were worth a little more than 10 cents a ton, and there were left in the 0.937 ton of solution only 0.146 gram each of gold and silver. Hence the result was as follows:

	Gold, grams.	Silver, grams.
Original content	5. 11	12. 7
Residual solution	. 146	. 146
Difference	4. 964	12. 554
Actually recovered in clean-up	4. 323	11. 215
Losses and assay samples	. 641	1. 239
Loss in tailing	. 146	. 146
	6. 787	1. 385

EFFICIENCY OF DEPOSITION.

The total time of the experiments during which the current was passing was 127.5 hours. The total number of ampere-hours was 126. The weight of gold actually recovered, divided by 7.35, the weight in grams of gold that should have been deposited in 1 ampere-hour, was as follows: $\frac{4.323}{7.35} = 0.588$ ampere-hour, which should have

* Compare with the 0.79 value given above.

* Agrees with the 1.48 value given above.

sufficed to deposit the amount of gold at 100 per cent electrical efficiency. The weight of silver actually recovered, divided by 4.025, the weight in grams of silver that should have been deposited in 1 ampere-hour was: $\frac{11.215}{4.025} = 2.79$ ampere-hours, which should have sufficed to precipitate that amount of silver at 100 per cent electrical efficiency.

Therefore a total of 3.378 of the 126 ampere-hours was actually utilized. Hence the total electrical efficiency was: $\frac{3.378}{126}$, or 2.68 per cent. This is a much higher efficiency than that of the Siemens & Halske process, and the capacity, being at the rate of 5 tons per 24 hours in a 1-ton box, is nearly double. The reason for the higher efficiency is that a shorter current gap is used, being only $\frac{1}{2}$ inch, as against $1\frac{1}{2}$ to 3 inches. In the Siemens & Halske process it is not possible to bring the electrodes near one another on account of the danger of short circuits from the loosely hanging lead-foil cathodes. It is probable that in practice I should have had to increase the distance between the electrodes, owing to corrosion of the anodes; but with peroxidized-lead anodes, provided with vertical separators of wooden strips, placed about 6 inches apart horizontally, better precipitation could have been safely obtained with even a $\frac{1}{4}$ -inch current gap.

CRITICISMS OF THE CHRISTY PROCESS.

In 1903 there appeared in Halle a small volume on "Cyanide processes of gold recovery," being one of the "Monographs on applied electrochemistry," edited by Viktor Engelhardt, chief engineer and chief chemist of the Siemens & Halske Co., Vienna. This particular book is written by Manuel von Uslar, with the assistance of Dr. Georg Erlwein,^a engineers of the Siemens & Halske Co., Berlin.

From this combination of electrochemists, representative of Siemens & Halske, much might have been expected, particularly as regards a description of the Siemens & Halske process; but although the process is mentioned, the description is meager and the fund of information that might have been reasonably expected is entirely missing. Concerning my process of "progressive electroconcentration" these authors remark:

Attention need only be called to the great sums that must be invested on the large scale in the form of sheet gold and silver to show that this patent can scarcely lay claim to practical consideration. As regards technical considerations it must be remarked that there are insuperable difficulties in continu-

^a Von Uslar, Manuel, and Erlwein, Georg, *Cyanid-Prozesse zur Goldgewinnung: Monograph. angew. Electrochemie*, 1903, Halle, Germany, 100 pp.

ously watching the electric current, so that anodes to be stripped of their gold (cathodes from bath 1) shall remain in bath 2 exactly till their gold covering shall be removed, which for a rational operation is quite necessary, as otherwise probable losses of gold, of current, power, and electrode material are unavoidable.

I am ready to admit many defects in the process as it appeared in 1900; these I shall refer to later; but the defects that require consideration are not those mentioned by the distinguished authors. No gold and silver sheets have to be provided except those formed in the ordinary course of the clean-up.

Of the Siemens & Halske process, Von Uslar and Erlwein* speak as follows:

Every month with electrolytic precipitation likewise the clean-up takes place, which, however, is simpler than with the zinc precipitation. Without interrupting the operation one lifts the single cathodes out of the bath in which a newly charged frame (cathode) is replaced, an operation which requires only a few minutes.

They then go on to describe the drying of the lead sheets, and the fluxing and smelting of the lead cathodes weighing 12 to 20 times the weight of the contained gold and silver, all of which has to be cupelled to recover the 6 to 8 per cent of gold. Anyone who has had any experience with the Siemens & Halske clean-up is well aware of the great danger of loss in handling the cathodes during and after the drying. The monthly clean-up withholds from circulation the same amount of gold and silver as any other process that requires a monthly clean-up.

With the Christy process, the cathodes, when charged, may be removed for stripping and be replaced by a clean cathode without interruption, as in the Siemens & Halske process. Moreover, with the Christy process, a weekly or even a daily clean-up is possible, so that it is not necessary to keep so much gold and silver in an unproductive state; neither must 12 to 20 pounds of lead be cupelled for each pound of gold, nor must the lead be reduced and rolled into sheets to be again used as cathodes.

As regards the second criticism of these authors, namely, that of a necessity of watching the current to determine exactly when plates are stripped, the criticism is not warranted. If 1 volt or less is used in the clean-up box the plates strip rapidly, do not corrode appreciably, and require practically no attention except removal, washing, and replacing in the deposition box for a new coating. If necessary, an electric bell could be devised to indicate when the plates were sufficiently stripped. When the solution in the clean-up box becomes rich, by raising the voltage the gold and silver are recovered by electrolysis much more easily than from the original solution.

* Von Uslar, Manuel, and Erlwein, Georg, *Cyanid-Prozesse zur Goldgewinnung*: Monograph. angew. Electrochemie, 1908, Halle, Germany, 100 pp.

Since writing the above statement I have seen an important paper by Neumann.* He remarks, "It is true Christy's proposition has some weak points, but certainly it does not fail for the reason given by Von Uslar and Erlwein." Neumann gives the results of some valuable experiments on the efficiency of electrical precipitation under the conditions of the Siemens & Halske process. However, he did not entirely reproduce the conditions, but kept the gold content constant by adding gold to the solution and, instead of keeping a constant voltage as in the Christy process, he maintained a constant current with increasing voltage. He seemingly did not attempt a complete precipitation as is necessary in practice. With a current density of 0.5 ampere per square meter, and with 10 grams of gold present per cubic meter, he found as a maximum efficiency 3.81 per cent; whereas, for the same density of current and with 3 grams of gold present per cubic meter, the efficiency is only 1.33 per cent. With a lower current density he got, for 0.25 ampere per square meter and 10 grams of gold per cubic meter, an efficiency of 7.56 per cent, and for 3 grams of gold per cubic meter, 3.15 per cent. For greater current densities such as 2.4, 4, and 9 amperes per square meter, and with 10 grams of gold present per cubic meter, he found the maximum efficiency to be 0.41, 0.24, and 0.16 per cent. It is evident that if the attempt had been made to reduce the gold content to a minimum the efficiency would have been lower.

Neumann points out what he claims to be the weak points of the Christy electroconcentration process. First, he claims that neither lead nor iron can be used alternately as cathodes and as anodes in cyanide solutions, as those metals are attacked by the cyanide and rendered unfit to serve as cathodes after having served as anodes in the clean-up cell. He fails to observe that the Christy patent states, "The anodes and cathodes may be made of any electroconducting material not too strongly acted on by the solution." Though lead and iron are mentioned, the Christy patent is not confined to their use. I have successfully used Acheson graphite sheets and other forms of carbon. I am ready to acknowledge that there is difficulty in the use of lead. However, the difficulty does not arise from the cause assigned by Neumann, but from local action if the electrodes are not used continuously and are put away damp.

Neumann quotes some experiments, made presumably under his direction, by Mr. John Johnston, in which gold was deposited on carbon cathodes after which an electric clean-up was attempted, supposedly with the Christy process. In the first four experiments,

* Neumann, B., *Electrolytic precipitation of gold from cyanide solutions*: *Electrochem. and Met. Ind.*, vol. 4, August, 1906, p. 300.

with a current density of 992 to 118 amperes per square meter, no gold at all was deposited. (Seemingly, the current deposited the gold in the form of slime, which was easily washed off by the violent agitation.) With 24 to 69 amperes per square meter the efficiency, 0.03 to 0.045 per cent, was low, and was higher with the lower density. The voltage, an important item, is not given. The stripping went on easily, but he forgets that the cyanide solution was strong—17.76 to 12.1 per cent. It is remarkable that any gold was precipitated from so strong a solution of free cyanide.

Examination has been made of samples of the silver and gold precipitated by the Christy process, the metal having been first deposited upon iron cathodes from weak solutions, then redissolved by making anodes of these cathodes in a 0.1 to 1 per cent cyanide solution, and reprecipitated upon sheet-iron cathodes, coated with a film of graphite and vaseline. The samples showed that it is possible to deposit coherent sheets of both gold and silver by the Christy process in exactly the way specified in the patent. After wrongly condemning the process as impossible, Neumann proposes to resort to carbon cathodes, which he would strip, causing the gold to be redeposited by the Wohlwill process, sodium chloride, gold chloride, and free hydrochloric acid being used as an electrolyte. By this means good results are obtained with pure gold, and a high efficiency results; however, upon calculating the gold as univalent the efficiency would be modified.

Mention should be made of the fact that the Wohlwill process is not applicable to alloys containing more than 10 per cent silver. With solutions carrying, as many do, more silver and copper than gold it is doubtful whether this ingenious modification of the Christy method would be applicable. The process could not be used to produce fine gold, as Neumann supposes, but by using cyanide, of a proper strength, and employing a proper voltage, the gold and silver could be recovered if the process were properly conducted. The operation is especially easy if cathodes composed of thin sheets of Acheson graphite are used as "cathode anodes." As pure cyanides free from sulphates and chlorides can be used in the stripping box, they are little corroded at the low voltage used in the Christy process.

SUBSEQUENT PATENTED PROCESSES.

C. P. Stewart, U. S. patent 676526, July 16, 1901.—The solution flows over a stationary mercury cathode. The process is similar to the "Molloy process."

William Orr, U. S. patent 689018, December 17, 1901.—Inventor uses zinc anodes in cyanide solutions, especially when the solutions contain copper, and regenerates the cyanide by throwing down the zinc by KHO and Na₂S. The patent resembles one granted to C. H. Merrill for regenerating cyanide from solutions containing zinc.

E. D. Kendall, U. S. patent 698292, April 22, 1902.—The specifications of this process show an electrodeposition box containing carbon anodes in porous cells. The remaining space of the outer vessel is filled with fragments of carbon, which act as cathodes upon which the gold and silver are deposited. After sufficient time the caustic potash is withdrawn from the porous cell and a cyanide solution, adequately strong, supplied to fill it. The carbon plate is then removed and a silver-plated copper plate is made the cathode in place of the anode. The gold and silver dissolve from the first carbon anode and deposit upon the inner plate. The invention contains ideas shown in the Christy patent (No. 643096) previously mentioned, but without the removable cathodes. In one place the inventor specifies a pressure of 5 volts and in another preferably 15 volts. It is evident that the porous cell is a disadvantage, as it increases the resistance enormously. I can not find any place where the process has been applied.

S. Muffy, U. S. patents 714598, 714599, November 25, 1902.—This patent is for a process and an apparatus. A carbon anode is used with a cathode composed of a filiform mass of lead and zinc.

THE BUTTERS PROCESS.

Charles Butters (U. S. patent 756211, Apr. 5, 1904) claims the use of a dense current in combination with peroxide-of-lead anodes and tin-coated steel cathodes. His claims are as follows:

First. The improvement in the process of precipitating metals from solutions, chiefly cyanide solutions, which consists in employing cathodes having surfaces of tin and a high-density electric current, substantially as described.

Second. The improvement in the process of precipitating metals from solutions which consists in employing cathodes having surfaces of tin in combination with anodes of lead peroxide and a high-density electric current, substantially as described.

Third. The improvement in the process of the electrolytic separation of metals from solutions which consists in using a cathode having smooth surfaces of tin, substantially as described.

In the specifications, Butters directs that the anodes and cathodes shall be placed about 3 inches apart, and that the solution containing the metal or metals to be deposited shall be caused to flow between the electrodes, preferably in an upward or downward direction, at a uniform velocity, while an electric current of high density enters the cathode. In practice about 0.5 ampere per square foot of cathode is found suitable, as a rule. The dissolved metals are then deposited in a loose slimy form on the tin cathode surface and may be removed by brushing or wiping the plates with a soft material, such as rubber or wood. This may be conveniently done by removing the cathodes from the solution when the deposit is sufficiently thick, or usually by

running a wiper over the plates at intervals of, say, once a day while in position in the solution, and allowing the removed slime to settle to the bottom of the vessel. This slimy deposit is removed from the vessel at regular periods, and the metals are separated and refined in the usual way.

It may be noted that in the Christy patent, issued in 1900, I disclose the use of a dense current with flat electrodes in the following words: "The current should be so graduated that it (the gold) is deposited in a firm, coherent coat. If a strong current is used, the gold is deposited on a cathode as a fine brown powder, which may be brushed off at intervals and melted down; but I prefer to deposit it as a firm, coherent coat to avoid mechanical loss in handling." Butters's process has been developed on a large scale in several of his plants in this country, Mexico, and Central America, and may be said to be technically successful, but there are at least two practical objections to it. The first is that so high a current density as he employs to separate the material in a loose form on flat electrodes is wasteful. The second is that the recovery of the precipitated slime by removing the plates and wiping them by hand involves much labor, and if the precipitate is scraped from plates and allowed to accumulate at the bottom of the tank there is not only a danger of loss, either by resolution or by mechanical means, but the precipitate is liable to be contaminated by the peroxide of lead that occasionally drops from the anodes to the bottom of the tank, or especially by the precipitated carbonate of lime which is continually settling out of the solution.

That this contamination of the gold is serious I have demonstrated by analyzing a precipitate coming from one of the Mexican plants where the process was used. The precipitate from the head of the boxes contained enough lead and oxide of lead to make it melt almost like butter, but the precipitate from the lower end of the tank contained so much carbonate of calcium that fusing was difficult. The bullion produced from each precipitate was necessarily of low grade. Nevertheless the process is in successful use. It is probably the only electrolytic process now in actual use for the treatment of cyanide solutions from gold and silver recovery.

EFFICIENCY OF BUTTERS PROCESS.

An admirable account of the Butters process is given by Hamilton.* The solution from the sand treatment has the following gold and silver content:

* Hamilton, E. M., A development in electrolytic precipitation of gold and silver from cyanide solutions: *Electrochem. Ind.*, vol. 2, April, 1904, p. 131.

	Gold, pennyweights.	Silver, ounces.
Head solution-----	2.93	3.51
Tail solution-----	0.26	0.29
Precipitated per ton solution-----	2.67	3.22

There were precipitated, at 3 volts, 216 tons of solution a day; hence there was recovered 898.56 grams of gold and 21,632.40 grams of silver. In depositing the gold there was utilized $\frac{898.56}{7.35} = 122.2$ ampere-hours. In depositing the silver there was utilized $\frac{21,632.40}{4.025} = 5,374.5$ ampere-hours, making a total of 5,496.7 ampere-hours. The area of cathodes was 6,950 square feet, with a current density of 0.55 ampere per square foot, or a total of 3,822.5 amperes; and in 24 hours we have 91,740 ampere-hours, or an electrical efficiency of $\frac{5,496.8}{91,740}$, which is slightly less than 6 per cent ampere efficiency. There was precipitated 91.1 per cent of gold, and 91.7 per cent of silver.

The solutions from slime treatment had the following content:

	Gold, pennyweights.	Silver, ounces.
Head solution-----	1.10	1.45
Tail solution-----	0.13	0.15
Precipitated per ton of solution-----	.97	1.30
	1.5085	40.4345

There was precipitated, at 2.6 volts, 480 tons of solution per 24 hours, and the recovery was 724.08 grams of gold and 19,408.56 grams of silver. Hence there were utilized in precipitating gold $\frac{724.08}{7.35} = 98.5$ ampere-hours, and in depositing silver $\frac{19,408.56}{4.025} = 4,822$ ampere-hours, making a total in useful work of 4,920.5 ampere-hours. The cathode area (like that of the anodes) was 13,536 square feet. The density of 0.3 ampere per square foot gives a total current of 4,060.8 amperes; or, of 97,459 ampere-hours, only 4,920.5 were utilized, and the electrical efficiency was only 5.05 per cent. The results are much better than those of Siemens & Halske, owing to the use of peroxide of lead instead of rusty iron for the anodes and of bright tin instead of lead for the cathodes. There was precipitated 88.2 per cent of gold and 89.6 per cent of silver. It should be remarked that there was precipitated simultaneously by the current an amount of copper of almost half the weight of the silver. This would considerably increase the electrical efficiency.

Taking the results quoted in Hamilton's figures for November and December, 1901, and for convenience converting ounces to grams by multiplying by 31.1, the figures are:

Total gold deposited in two months.....	Grams. 84, 872
Total silver deposited in two months.....	2, 204, 772
Total copper deposited in two months.....	1, 024, 030
For gold, dividing by 7.35, the number of ampere-hours utilized is.....	11, 547
For silver, dividing by 4.025, the number of ampere-hours utilized is.....	547, 769
For copper, dividing by 2.372, the number of ampere-hours utilized is.....	431, 716
Total (supposing copper to be univalent) number of ampere-hours utilized is.....	991, 032

During this time there was actually used 11,013,640 ampere-hours; hence the efficiency was 9 per cent. If, however, as Hamilton assumes, the copper was bivalent 432,000 ampere hours should be added for the copper, making a total of 1,423,032, or an efficiency of 12.9 per cent. Either result is a great improvement on that by the Siemens & Halske process.

In different experiments with various tonnages of these solutions, Hamilton has obtained efficiencies of 4.83, 6.10, 13.24 and 13.90 per cent, but he has not calculated the average. He has used practically the equivalents that I have taken for gold and silver, but for copper a decidedly lower one, being 1.206, as against the 2.372 which I used for univalent copper.*

It is altogether probable that copper, like gold and silver, is univalent in cyanide solutions; moreover, cupric cyanide is a very unstable compound. Gmelin and Hittorf both claim that copper dissolves only as cuprous cyanide. I have not investigated the subject, but have reason to believe that, at the anode, copper cyanides are formed which are of higher valence than 1, and that these compounds are subsequently reduced to others of lower valence at the cathode. This is one of the reasons that copper cyanide is so hard to precipitate by electrolysis.

STUDY OF CONDITIONS GOVERNING ELECTROLYTIC DEPOSITION.

It must be evident that the results obtained at Bodie were good, so far as precipitation is concerned. However, in 24 hours, with the box used not more than 5 cubic feet of solution could be treated per cubic foot of box capacity. The maximum duty of the deposition box would be at the rate of 5 tons a day for a box holding 1 ton of

* Standard Handbook for Electrical Engineers, 1915, pp. 1560-1564.

solution. This rate did not seem to me to be satisfactory, yet when I attempted to increase the velocity of the solution above this rate, a poor precipitation resulted.

While I was making these experiments at Bodie, it occurred to me that a good way to increase the capacity of the box would be to circulate the solution by pumping it repeatedly through the box, thus concentrating the gold and silver upon a smaller number of electrodes than would be necessary if the box were made large for complete precipitation by one passage. I then determined to make a systematic study of all elements necessary to procure complete and rapid precipitation of gold and silver by electrolysis. I had long investigated the effect of the current with sheet-iron electrodes upon a solution of cyanide of potassium at rest. Into this solution I immersed a sheet-iron anode $12\frac{1}{2}$ by 13 centimeters in size, and a sheet-lead cathode of about the

same dimensions, spacing these $1\frac{1}{4}$ inches apart. I used a storage battery of approximately 2 volts at first, and after $2\frac{1}{4}$ hours raised the voltage to 4 volts. The resulting ampere curves and the cyanide curves are shown in figure 1.

The experiment lasted 45 hours. At the end of the period the cyanide was reduced to 0.03 per cent, showing that 97 per cent had been destroyed. It will be seen that the current diminished almost immediately, owing to polarization effects and oxidation of the

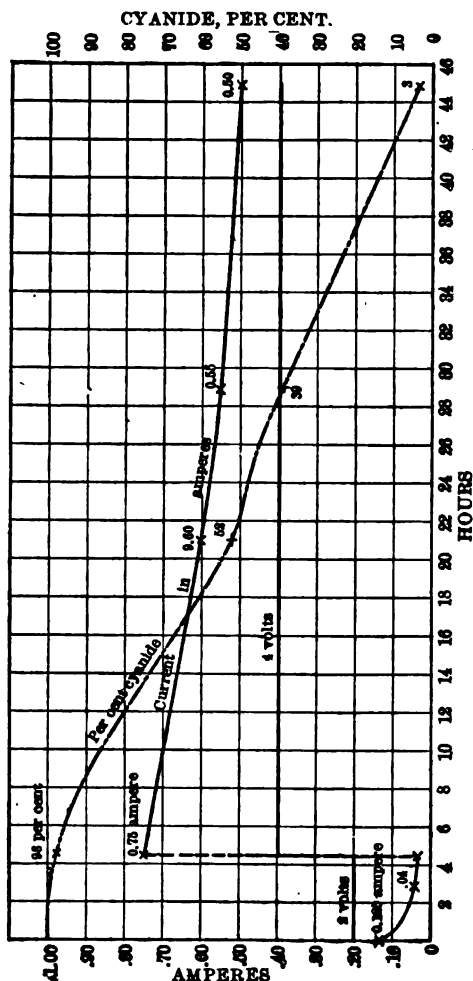


FIGURE 1.—Ampere-cyanide curves from experiments on effect of electrolysis on solution of cyanide potassium at rest. Volume of solution, 500 c. c.; KCy, 1 per cent; iron anode, $12\frac{1}{2}$ by 13 cm.; lead cathode, $12\frac{1}{2}$ by 13 cm.; separation, $1\frac{1}{4}$ cm.

anode. The anode was only slightly oxidized, an almost imperceptible film of oxide of iron having formed.

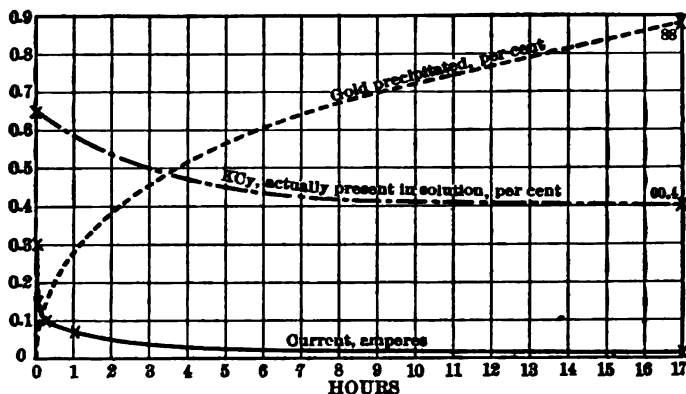


FIGURE 2.—Curves from experiments on electrodeposition of gold from solution of cyanide of potassium. Iron anode, 12½ by 13 cm.; lead cathode, 12½ by 13 cm.; current, 2 volts from storage cell; KCy, 0.065 per cent; Au, 0.005 per cent; volume of solution, 500 c. c.; Au, 2.5 mg.—\$3 a ton; electrodes 1½ cm. apart.

Following these tests and in the same apparatus a cyanide solution was treated containing 0.065 per cent of free cyanide and \$3 a

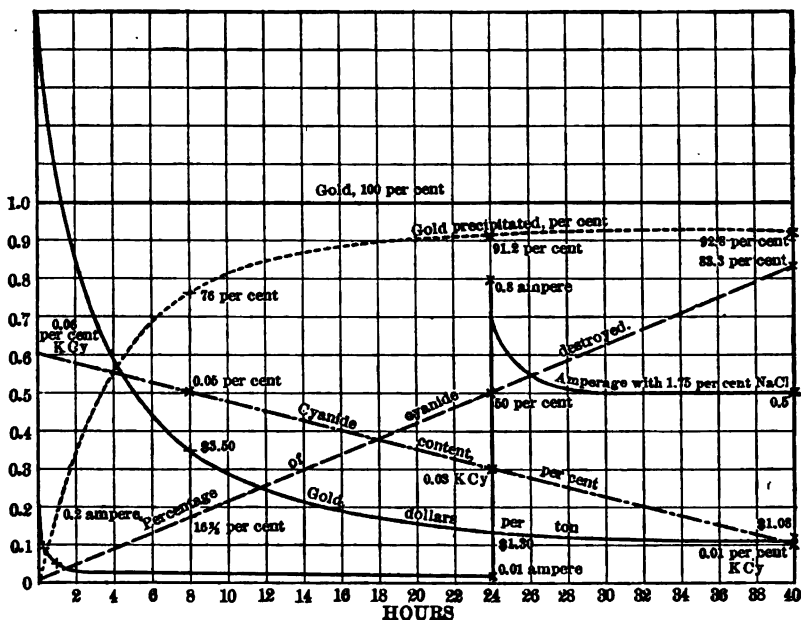


FIGURE 3.—Curves from experiments on electrodeposition of gold from solution of cyanide of potassium. Iron anode, 12½ by 13 cm.; lead cathode, 12½ by 13 cm.; electrodes 1½ cm. apart; current, 2 volts from storage cell; volume of solution, 500 c. c.; Au, \$15 a ton; KCy, 0.065 per cent.

ton in gold. The results of the experiment showed that in 17 hours only 88 per cent of the gold was precipitated. The cyanide was con-

tinuously destroyed, only 0.04 per cent remaining at the end of the experiment. The results of this test are shown in figure 2.

In the same apparatus a 0.06 per cent cyanide solution containing \$15 a ton in gold was treated. One storage cell was used and the experiment lasted for 40 hours. The results are shown in figure 3.

At the end of 24 hours the gold was reduced from \$15 to \$1.30 a ton. One-half the cyanide then had been destroyed. At this time I added sodium chloride, a suggestion of Mr. Butters, so as to make a solution containing 1.75 per cent NaCl. The current jumped immediately from about 1 milliampere to 800, and at the end of 40 hours remained 100 milliamperes. The gold had been

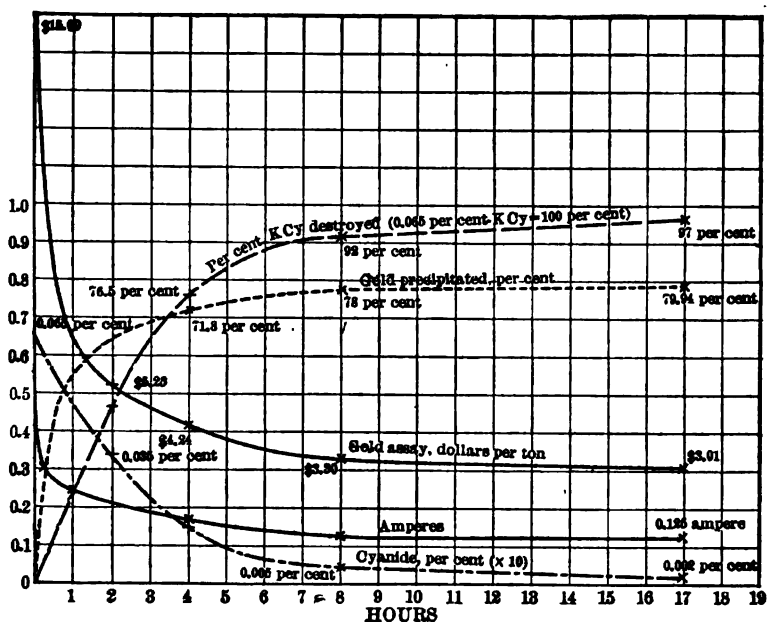


FIGURE 4.—Curves from experiments on electrodeposition of gold from solution of cyanide of potassium. Iron anode, $12\frac{1}{2}$ by 18 cm.; lead cathode, $12\frac{1}{2}$ by 18 cm.; electrodes $1\frac{1}{2}$ cm. apart; current, 4 volts from two storage cells; Au, \$15 per ton; KCy, 0.065 per cent.

reduced to \$1.08 a ton, and the cyanide to 0.01 per cent, so that only 83.3 per cent of the gold was precipitated. Five-sixths of the cyanide had been destroyed. The low rate of precipitation in this experiment led me to increase the voltage to 4 volts, using two storage battery cells. For this I used an 0.065 per cent cyanide solution containing gold to the value of \$15 a ton, the remaining conditions being as before. The results are shown in figure 4. At the end of 17 hours the cyanide had been reduced to 0.002 per cent, 79.94 per cent of the gold had been precipitated, and 97 per cent of the cyanide had been destroyed. The results did not then look promis-

ing, but when compared with the results of my later experiments, they show the great progress that has been made in this investigation.

One of the first things to be noted in the results is the rapid fall of the current, which continued to decrease throughout the whole process. I noticed that whenever either of the electrodes was agitated so as to displace the solution in immediate contact, the current immediately increased, falling again as soon as the solution came to rest and the conditions were reestablished by electrolysis. This was evidently due to the well-known phenomena of polarization. It was made evident by these experiments that the precipitation of gold and silver from cyanide solutions was a problem of great difficulties.

HOW DOUBLE CYANIDES REACT IN ELECTROLYSIS.

The solutions of the double cyanides seemingly act much differently from those, for instance, of either sulphates or chlorides of a metal like copper. With solutions of these latter salts rapid and complete precipitation would at once ensue. To those familiar with the conditions of electroplating gold and silver in cyanide solutions, the electrodeposition process seems to be a natural and easy one; but the conditions there maintained are different from those prevailing in solutions used in ore treatment. With solutions such as are used in electroplating the density is sufficient to produce a very good conductor. The solution is rich in gold or silver, and the metal deposited is replaced as fast as it is precipitated by a supply from the soluble anodes. Hence the electrochemical conditions are suitably maintained. However, in the treatment of solutions resulting from ore treatment, solutions have to be handled that assay less than \$6 per ton, and contain less than 0.001 per cent of metallic gold. This solution must, perhaps, be reduced to less than 6 cents per ton, or 0.00001 per cent of gold. No one would attempt such a task with a metal less valuable than gold or silver.

The first part of the gold in electrolysis is readily removed, but the deposition becomes increasingly difficult toward the end, and it is theoretically and practically impossible to remove the last traces of the gold from such solutions. The chief cause of the great difficulty is that the gold and silver form what are known as "complex ions" in the cyanide solution. For instance, if a solution containing pure KAuCy_2 but no free cyanide be considered, one is dealing with a substance which crystallizes in white crystals, looking very much like granulated sugar, soluble in water and forming a colorless solution. This solution, if sufficiently dilute, according to the ionic theory breaks up into two ions, the simple ion K carrying a positive charge of electricity, and the complex ion AuCy_2 carrying a negative

charge. When an electric current is passed through such a solution, with platinum electrodes, the potassium ion carrying the plus charge of electricity travels to the negative pole, and if there is sufficient gold cyanide in the vicinity it decomposes some of the cyanide, precipitating the gold on the cathode and setting free the cyanide. Meantime the complex ion AuCy_2 , with its negative charge of electricity, travels to the anode, and there forms a pale yellowish, slimy film which protects the anode from further action by the current. Hence, with such a solution, electrolysis soon comes to a stop, the anode being insulated by the slimy coat of AuCy_2 .^a As was proved by Hittorf,^b the same action takes place with potassium argentocyanide, the only outward difference being that the slimy film of AgCy_2 is white.

Because of the formation of the insoluble and insulating slimes on the anode it is always necessary, in electroplating, to have some free cyanide of potassium in the solution to redissolve this film. Caustic potash may be used instead, but is not so satisfactory. For instance, AuCy_2 partly decomposes into AuCy and Cy . The Cy partly oxidizes to cyanate and partly changes to cyanide of potassium. AuCy redissolves in the excess of free cyanide if present, or in KHO if only the latter is present. With the KHO , cyanate and cyanide of potassium are formed and the essential salt of the solution is regenerated. This rather complicated process is hard to understand and a little difficult to explain, but there is no doubt about its occurrence. As regards silver cyanide the reactions mentioned were demonstrated quantitatively by Hittorf.

In order to appreciate the importance of this peculiarity of cyanide solutions, consideration might be given to the electrolysis of nitrate of silver. The Ag goes directly to the cathode, and NO_3 travels to the anode. Hence for every atom of silver precipitated at the cathode one atom disappears from the solution near the cathode. But in the electrolysis of cyanide of gold or silver, for every atom of gold or silver precipitated at the cathode two atoms are removed by the current, one being precipitated by the K ion and the other traveling to the anode. Hence it must be evident that the action of the current renders further precipitation of the gold more and more difficult. Julian and Smart^c claim that this discovery was made by Daniell and Miller. They say—

Daniell and Miller (*Philos. Trans.*, 1844, p. 1) investigated the nature of the ions, and found that when an electric current is passed through a solution of KAuCy_2 or KAgCy_2 , the K is the positive ion and goes to the cathode, while

^a See Christy, S. B., On the solution and precipitation of cyanide of gold: *Trans. Am. Inst. Min. Eng.*, vol. 26, 1896, p. 758.

^b Hittorf, W., *Über die Wanderungen der Ionen während der Elektrolyse*, 1859; reprinted in *Ostwald's Klassiker der exakten Wissenschaften* No. 23, 1891, p. 74.

^c Julian, H. F., and Smart, Edgar, *Cyaniding gold and silver ores*, 1904, p. 121.

the negative ion consists of gold or silver and cyanogen, and this goes to the anode. Hittorf has since confirmed this and assumes the negative ion to be AuCy_2 for gold and AgCy_2 for silver. That is to say, that when these complex salts are electrolyzed, the gold and silver move in the opposite direction to that taken in the case of simple salts.

The foregoing abstract quotation appears to contain several mis-statements: In the first place, nowhere in the reference given do Daniell and Miller make any reference to the cyanides of gold or silver. They do discuss potassium ferro and ferri cyanide, cyanide of potassium, and sulphocyanide of potassium, but nowhere in the article mentioned nor in any other of the Philosophical Transactions from 1842 up to 1860, do they make any statement that can be twisted into such an assertion. They state that they intend to investigate other cyanides, but they do not seem to have done so.

It may further be remarked that the work of Hittorf seems to have been both underrated and overrated. No references by Julian and Smart are given to Hittorf's^a work, a full account of which has been reprinted by Ostwald. The work of Daniell and Miller was of great importance to electrolytic science, but it contained numerous errors which were cleared up by the skillful work of Hittorf which is a model of scientific accuracy. Hittorf with punctilious care seems to have given full credit to Daniell and Miller for the good work done by them, but there is no mention of their having done any work with the cyanides of gold and silver. He seems to have been the first to have shown that in the electrolysis of a KAgCy_2 solution the K goes to the cathode and the AgCy_2 to the anode. However, he makes no mention of KAuCy_2 solution, though he does treat of HAuCl_4 .

Ostwald^b has collected data on all the known velocities of ionic transfer, and he gives the values for K and $\text{Ag}(\text{CN})_2$ and credits the work to Hittorf. Nowhere is any mention made of K and AuCy_2 , though the list covers eight pages. Hence I assume that Julian and Smart have claimed too much for Daniell and Miller. Whether I was the first to call attention to the nature of the electrolytic reaction in solutions of KAuCy_2 ,^c I do not know; but I believe that I can fairly claim to have been the first to bring out the practical importance of it in electrodeposition from cyanide solutions. The reaction is of controlling importance in all electrocyanide processes and yet it is not understood by many of the writers on the cyanide process or by many authors of chemical textbooks.

^a Hittorf, W., *Über die Wanderungen der Ionen während der Elektrolyse*, 1859; reprinted in Ostwald's *Klassiker der exakten Wissenschaften* Nos. 21 and 23, 1891, 87 and 142 pp.

^b Ostwald, Wilhelm, *Chemische Energie*, 2d ed., 1898, p. 608.

^c See Christy, S. B., *The solution and precipitation of cyanide of gold*: *Trans. Am. Inst. Min. Eng.*, vol. 26, 1896, pp. 758-759.

Sir Humphrey Davy's original idea as to the composition of salts as against the ideas of his time has been confirmed by more recent scientific investigations. Daniell and Miller do not seem even to have been the first to discover the complex ions of potassium ferrocyanide, ferri-cyanide, and sulpho-cyanide. To Hittorf belongs the credit of expanding and completing the application of the results of early investigators to potassium silver cyanide. Ostwald deserves the credit of generally clearing up the subject by the invention of the term "complex ion." As far as I am aware he did not extend his work to include KAuCy_2 .

The separate experiments upon which I based the statements made in 1896 were with cyanides of silver, of gold, and of copper. I used pure solutions of the double salts of potassium and of the several metals without free cyanide. A platinum dish was the cathode, a platinum wire the anode. I found that the current was soon stopped by the formation of a nonconducting slimy coating of varied color. With KAgCy_2 the coating was of a pale white appearance and had a faint yellow cast. The slime from the KAuCy_2 was yellow with a pale green cast. With copper the anode slime was a bright grass green or yellowish green. The accumulation of any one of these precipitates gradually stopped the electric current. When the crust of precipitate was scraped off the anode, so as to fall on the cathode, it retained its shape but was reduced to the metal—silver, gold, or copper—according with the composition of the double cyanide. Similar crusts form on the anodes in electroplating when insufficient free cyanide is used, as is well known to all electroplaters, but I have not been able to find the explanation of this phenomenon in any of the many books on electroplating published before 1896.

Fortunately a comprehensive bibliography on this subject has been prepared by McBain.^a It is claimed to be complete to the end of the year 1905. All the work of Daniell is included in this outline, and no mention is made of his ever having determined the migration velocities of KAgCy_2 or of KAuCy_2 . The relative velocity of the potassium ion in KAgCy_2 was determined to be 0.594 in a solution containing 1 gram-molecule in $1\frac{1}{2}$ liters.^b Nobody seems to have determined the migration velocities of the ions of KAuCy_2 up to the end of 1905.

Unfortunately at variance with the statements made by Julian and Smart, the fact appears that Daniell was not the first to observe the true nature of electrolysis of complex ions.

^a McBain, J. W., *Experimental data of the quantitative measurements of electrolytic migrations*: Wash. Acad. Sci., vol. 9, July 31, 1907, pp. 1-78.

^b Hittorf, W., *Über die Wanderungen der Ionen während der Electrolyse*, 1859; reprinted in *Ostwald's Klassiker der exakten Wissenschaften* No. 23, 1891, 142 pp.

The first observer seems to have been Robert Porrett in 1814.^a The paper mentioned is signed "Tower, June 6, 1814." It would appear from the large number of French chemical terms used that Mr. Porrett must have been a Frenchman, probably residing in England, and not a member of the Royal Society.

Porrett seems to have had a correct idea of the composition of the salts known as triple prussiates, and to have fought the idea then prevailing that the triple prussiates were simple, double cyanides. Thus, he says:

I consider the salts termed triple prussiates as binary compounds of an acid with a single base; as salts which do not contain any prussic acid, nor any oxide of iron as a base, although both the substances may be obtained from them by a decomposition of their acid.

The first experiment which I shall adduce in support of the above opinion is one with the voltaic battery; it appeared to me that this instrument would show whether the oxide of iron in the triple prussiates existed in them as a base, or as an element of a peculiar acid by attracting it to the negative pole in the former, and to the positive pole in the latter case. I therefore exposed a solution of triple prussiate of soda to the agency of a small battery of 50 pairs of double plates of one inch and a quarter square, kept in action for 20 hours, the solution was connected by platinum wire with the negative pole, and by filaments of cotton with distilled water which communicated by platinum wire with the positive pole. Thus circumstanced the triple prussiate of soda was decomposed, its acid (consisting of the elements of prussic acid and black oxide of iron) was carried over to the positive pole; here meeting with oxygen from the decomposing water, it underwent a farther change by which it was converted into prussic acid which was partly volatilized, and into blue triple prussiate of iron which formed there in abundance; the liquid at the negative pole after this process contained only soda with a trace of undecomposed triple prussiate. In this experiment I consider the circumstance of the black oxide of iron being carried over to the positive pole as a proof that it went there as an element of an acid, for as a base it must have remained at the negative pole.

Porrett seems to have discovered also sulpho cyanate of potassium (which he calls "red-tinging acid") by boiling Prussian blue with sulphide of potassium, and also by heating to redness animal charcoal and sulphide of potassium, as well as by other means. He subjected this solution to the action of electrolysis and found that the new acid was carried to the positive pole without decomposition or the separation of any sulphur. This would clearly prove, it seems to me, that to Porrett belongs the credit of first proving the reaction that takes place when "complex ions," as they are now termed, are electrolyzed.

The first quantitative work on "transfer values" seems to have been done with sulphate of sodium by Faraday^b in 1833. Daniell's

^a Porrett, Robert, jr., On the nature of the salts termed triple prussiates, and on acids formed by the union of certain bodies with the elements of the prussic acid: *Phil. Trans. Roy. Soc. Lond.*, 1814, pp. 527-556.

^b Faraday, Michael, Experimental researches in electricity: *Phil. Trans. Roy. Soc. Lond.*, 1833, pt. 1, p. 36.

first work was reported in 1839, and his last work seems to have been done in 1844, whereas Hittorf's work was from 1853 to 1859.

The theories foreshadowed by Sir Humphrey Davy, proved to be true by Porrett and by Faraday, afterwards confirmed by Daniell and Miller, and carried to perfection in Hittorf's classic work, have been of the greatest importance not only to chemical science but to practical technology. The work of these men, not even yet fully appreciated, was for many years a subject of bitter controversy.

PRACTICAL APPLICATION OF THE LAW.

It soon became evident to me that the necessary electrolytic reaction with the double cyanides was one of the chief causes of slow precipitation, in short, that the gold content at the cathode was so rapidly diminished that the potassium ion when brought to the cathode by the electric current found there no gold to precipitate. Consequently, electrical energy was wasted, water being decomposed, and caustic potash and hydrogen were formed to no useful purpose. The gold was accumulated at the anode where it acted injuriously by increasing the resistance.

It occurred to me that both these evils could be at once met by a rapid circulation of the solution from anode to cathode, so that the gold collected at the anode, where it was doing only harm, could be swept to the cathode where the potassium ions, instead of decomposing water, would precipitate gold. With sheet electrodes it was not easy to obtain this result, whether, as in my first process, the electrodes were transverse to the box, and the circulation up and down between them, or whether, as in most recent installations of the Siemens and the Butters processes, the electrodes hung longitudinally in the box, and the solution passed directly between them. With none of these arrangements could the direct replenishment of the cathode solution, so necessary to good results, be effected. This requirement could be met only by the use of pervious anodes and cathodes, either of sheet metal or of wire cloth. Perforated sheet electrodes were first used but the cyanide of gold and silver accumulated on the anode spaces, between the holes, where the circulation was poor. I then resorted to the use of iron-wire cloth.

The first anodes were made of iron-wire cloth of 8 meshes to the linear inch, the wires being 1 mm. thick. A single sheet was used which was $3\frac{1}{4}$ inches wide by 4 inches high, only about $2\frac{1}{2}$ inches being submerged. It was stiffened by a thin, sheet-iron fold having a copper conductor riveted to the top. The first cathodes were of similar dimensions and were made of one-fourth-millimeter iron-wire cloth of 30 meshes to the linear inch. I afterwards found that such cathodes offered too much resistance to the current and finally made them of about one-half-millimeter wire cloth of 16 openings to the linear inch. Such a piece of wire cloth has almost exactly the

same area for the submerged wires as would be presented by a solid sheet of metal of the same cross section exposed on both sides.

It is well known that when oxidation is desired in electrolysis a large anode area should be used. When reduction is wanted a large cathode area should be used. As reduction rather than oxidation was the purpose of the process it seemed wise to have a larger area of cathode than of anode. Hence, five such cathode sheets were fastened together and brought into contact with a copper conductor, above the water level, by crimping them at the top by a fold of sheet iron and at the bottom by similar small sheet-iron crimps. This makes what I term herein "bunches of five." After various trials I found that two such "bunches of five" cathodes, ten sheets in all, could be effectively inserted between each pair of anodes. The metal then deposits on all of the ten, most of course on the sheets next to the anodes, but upon all in time. These cathodes were three-sixteenths inch thick at the crimps and one-quarter inch at the widest part. The anodes were insulated from the cathodes by vertical rubber bands one-eighth inch thick. This arrangement was compact, giving over five times as much cathode as anode area. Since wire cloth, as manufactured, is coated with a protective coating of oil, it was always necessary to burn this off at a red heat before constructing the electrodes. To avoid undue oxidation, this burning must be done with care, particularly when the wire cloth is to be used for cathodes. A slight film of black magnetic oxide on the anodes is an advantage rather than a detriment, as it partly protects them from attack by the solution. With solutions free from salt and soluble sulphates the iron anodes are hardly attacked. Some of these anodes and cathodes that have been in use at intervals for 14 years are still serviceable.

Another advantage of the wire-cloth cathodes is that a dense current can be used without forming a loose deposit, and hence a rapid deposition can be obtained. The deposited metal tends to form about the wire a tube, which adheres firmly. This has long been recognized in the electrolytic assay of copper, where the use of platinum-wire gauze has so much reduced the time necessary for complete precipitation.

The advantages of the use of wire-cloth cathodes were as follows:

1. Direct and effective circulation of the solution from anode to cathode.
2. Larger area of cathode than of anode.
3. Use of a denser electric current without waste.

All of these advantages were not appreciated at once, but became gradually of more apparent importance as the investigation went on. The first experiments were tentative, and though the methods used were rather crude, compared with later ones, I give them rather fully because of the importance of the details.

PRELIMINARY EXPERIMENTS.

On September 17, 1900, I took a box $28\frac{1}{2}$ centimeters long by 6 centimeters deep and 7 centimeters wide, and holding 1.195 liters (0.0422 cubic foot) of liquid. In this were placed eight anodes made of sheets of 8-mesh iron-wire cloth, of wires 1 mm. in diameter. The nine cathodes were made of iron-wire cloth of 30 meshes to the linear inch. The immersed area of the cathodes was $6\frac{1}{2}$ square inches, or 0.0476 square foot. The actual surface of the cathodes was approximately double that area, or $13\frac{1}{2}$ square inches (0.0952 square foot). I used 8 liters of solution of 0.2 per cent KCy, assaying 15.87 mg. of silver per 100 c. c.

At 3.40 p. m. the first experiment was started, the current being 0.20 ampere, at 1.1 volts, and the solution containing 0.19 per cent KCy and 15.87 mg. of silver per 100 c. c. The source of current was a storage battery of 2 volts on an open circuit. Some resistance was used in series with the battery to adjust the voltage. At 5 p. m. the solution had passed through the box once, and the current was found to be 0.16 ampere at 1.25 volts, the solution containing 0.18 per cent KCy and 5.61 mg. of silver per 100 c. c. The recovery in the one passage through the box was therefore 64.50 per cent.

During this experiment no evolution of gas at the cathodes was visible or audible. After the first run had been finished, 8.4 c. c. of nitrate of silver, or 0.084 gram of silver, got into the solution by accident. In order to neutralize this, a proper proportion of cyanide of potassium was added to the solution. This increased the original assay value of the solution to 16.98 mg. of Ag per 100 c. c. The solution was then left at rest over night, with the battery connected. The voltage rose to 1.75, and the current fell to 0.06 ampere, and by morning the content of Ag had been reduced to 0.30 mg. per 100 c. c.

At 9.25 a. m. the solution was started a second time through the box. The voltage was 1.75 volts and the current 0.10 ampere. The solution had all passed through at 1.30 p. m. The silver had been reduced to 0.74 mg. per 100 c. c., showing that there had been precipitated, in the first and second run, 95.64 per cent of the original content.

At 2.10 p. m. the third run was begun at 1.75 volts and 0.2 ampere. The run was finished at 2.50 p. m. The solution contained 0.16 per cent KCy and 0.54 mg. of silver per 100 c. c. The recovery was 96.82 per cent.

At 2.55 p. m. the fourth run was started at 1.8 volts and 0.35 ampere. The solution contained 0.16 per cent KCy and 0.29 mg. of silver per 100 c. c. The recovery was 98.29 per cent.

At 3.18 p. m. the fifth run was started and was finished at 3.35 p. m. The solution contained 0.16 per cent KCy and 0.20 mg. of silver per 100 c. c. The recovery was 98.82 per cent.

At 3.39 p. m. the sixth run was started and was finished at 5.14 p. m. The solution contained 0.15 per cent KCy and 0.09 mg. of silver per 100 c. c. The recovery was 99.47 per cent.

A résumé of the results during the total period of 8.3 hours follows:

Results of early experiments with electrodeposition of silver.

Run No.	Duration of run.	Recovery.	Silver remaining in residue.
	<i>Hours.</i>	<i>Per cent.</i>	<i>Per cent.</i>
1.....	1.55	64.50	35.50
2.....	4.08	95.64	4.36
3.....	.66	98.82	3.18
4.....	.40	98.29	1.71
5.....	.28	98.82	1.18
6.....	1.55	99.47	.53

Some solution was consumed in the samples, but, making no allowance for this, 8 liters of solution was treated, the silver content being reduced to 0.53 per cent in 8.30 hours. The box held 1.2 liters of solution, but the electrodes occupied somewhat less than 1 liter of this space, so that the solution was treated at the rate of about 1 liter per hour in a 1-liter box, the silver recovery being 99.47 per cent. Thus the capacity of the box, with wire-cloth electrodes, was 24 liters per 24 hours, or nearly five times the capacity with the plain electrodes used in my experiments at Bodie. The solution of 8 liters passed through the box six times in the 8.3 hours, or roughly speaking, it required 1.4 hours for 8 liters to pass once, and hence it required on an average 1.75 hours for 1 liter to pass through the box. The average rate of flow through the box was 5.7 liters per hour.

CIRCULATION OF SOLUTION.

The importance of a proper rate of flow through the box is evident from the following notes taken during the first run:

12.30 p. m.—Volts, 1.60; amperes, 0.120; solution in circulation.

12.45 p. m.—Circulation stopped; voltage at once raised to 1.73; amperes fell to 0.070.

1.00 p. m.—Volts, 1.74; amperes, 0.068; solution at rest.

1.30 p. m.—Volts, 1.75; amperes, 0.060; solution at rest.

2.10 p. m.—Solution started through the box for the third time; volts, 1.75; amperes, 0.200.

In all the experiments, when the circulation of the solution was stopped, the current dropped, and the voltage rose. The solution was made to circulate by hand, and after having passed through the box once it was placed again at the head and allowed to flow again, the rate being regulated by the stopcock. The results from the first run indicated that I was on the right track to increase the capacity of

the box, and the subsequent experiments were undertaken to determine the best conditions.

In all my first experiments the circulation was made by hand, as described; but it soon became evident that a rapid rate of circulation was so important that some automatic means ought to be used. For the purpose a small crude centrifugal pump like that shown in figure 5 was made.

A number of persons have tried to verify my results on a small scale without using a proper means of circulation. Centrifugal pumps as small as that shown are not to be purchased and it is

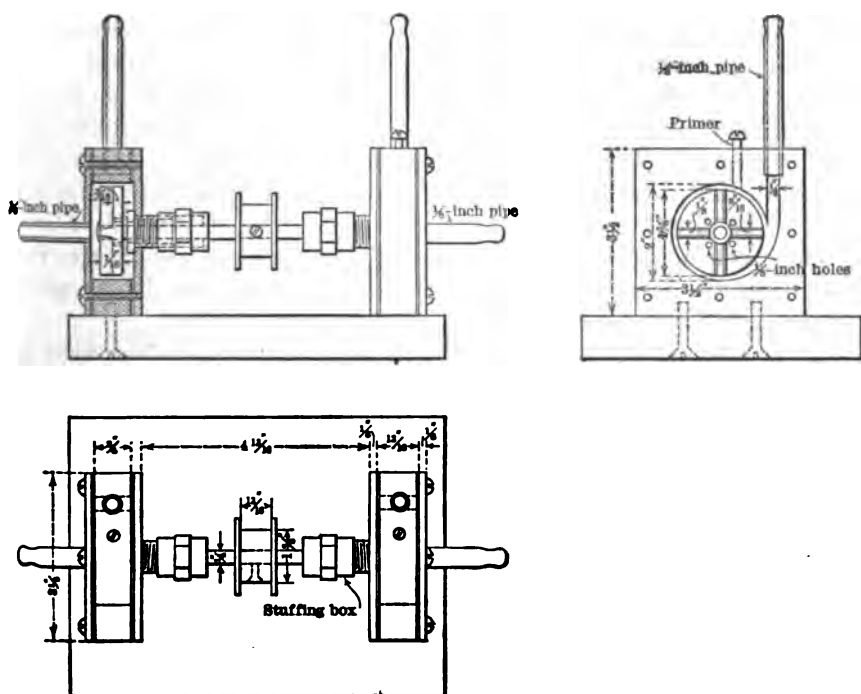


FIGURE 5.—Small double centrifugal pump used in circulating solutions.

necessary to make them, but they need not be elaborate or costly. The small double centrifugal pump shown can easily be constructed in any machine shop at small expense. It gives ample circulation for electrodes 10 by 10 cm., or 4 by 4 inches. This size is most convenient, as it has a sectional area of one one-hundredth square meter or one-ninth square foot, and results can be converted into square meters by multiplying by 100 or into square feet by multiplying by 9.

Two pumps are easier to use than one because they run in better balance (see fig. 5). The pumps rest in a planed base of cast iron 1

inch thick. The body of the pump is a square block of cast iron three-fourths inch thick, planed true and faced on each side with graphited rubber gaskets. In the center of this plate a 2-inch hole is provided. The sides are of thin plates of iron, the outside ones carrying a one-eighth-inch suction pipe having the end turned for a hard-rubber hose attachment. The inner side carries a bearing, with stuffing box. The whole is secured to the inner plate by a lock nut. The packing is well lubricated with a mixture of vaseline and graphite, and should be adjustable to prevent leakage. It is important that the shaft through the stuffing box should be perfectly smooth so as not to cut out the packing. A pulley of thirteen-sixteenths-inch face, slightly crowning, and of $1\frac{1}{8}$ -inch diameter, is driven by a small electric motor or a Pelton water wheel. The runner is constructed with four blades and commonly should be placed in a position to be slightly eccentric, and tangent only near the discharge. The position of tangency most desired is not indicated in the particular pump here shown. The clearance elsewhere should be slight. It is important, too, to drill at least four one-eighth-inch holes through the web of the runner to allow it to run in balance. A one-fourth-inch drill hole leads the discharge to a one-eighth-inch pipe with a hose connection at the end.

It is astonishing how important a small pump of this kind is. Hand circulation is impracticable. Air lifts produce an aerated solution that tends to dissolve the gold. The pump shown, easily and cheaply made, has sufficed for thousands of experiments and is still in good order. Of course, for work on a large scale, a well-designed centrifugal pump would be necessary.

EFFECTS OF INCREASING THE RATE OF FLOW.

Most of the following experiments were conducted with solutions of pure silver potassium cyanide (KAgCy_2), which was cheap and convenient. The solutions were highly concentrated and were afterwards diluted to form the free cyanide solution desired, to which also other pure chemicals such as KHO , or CaH_2O_2 , and KCy were added as necessary. When proper precipitation had been procured the silver content was again raised by adding the proper volume of the strong solution, after which the process was continued. Only in this way could the many variable factors be kept under control. By the use of the rigid electrodes and of proper insulating spaces the anodes and cathodes could be brought within one-eighth inch of one another with perfect safety. A more compact construction is well nigh impossible.

At first I gradually increased the rate of flow and found that the capacity of the box increased proportionally. The rate was first

increased to 8 liters per hour, being subsequently increased to 10, 30, 60, and finally 120 liters per hour. The space occupied by the electrodes was about 1 liter. Also it was found convenient to increase the volume of the solution to about 22 liters. The results of a number of tests follow:

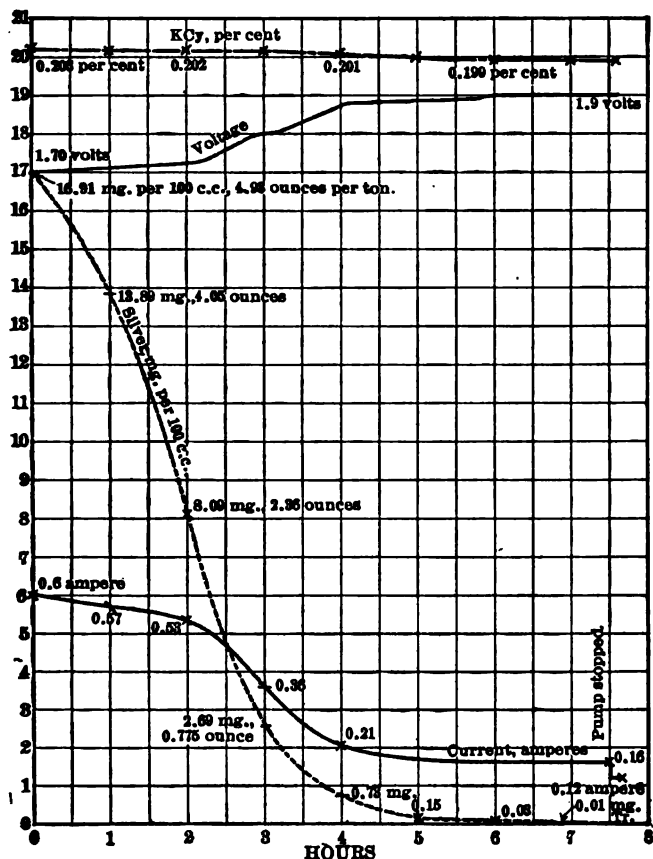


FIGURE 6.—Electrodeposition results with silver cyanide solution with rate of flow of 60 liters per hour; velocity, 9.4 inches per minute through open box, 21 inches through cathodes; 22 cathodes of 30-mesh iron wire, each $2\frac{1}{2}$ by $2\frac{1}{2}$ inches; area, 6.469 square inches; actual wire surface, 1 cathode, 0.1147 square foot, 22, 2.5234 square feet; estimated effective surface, 1 cathode, 0.1 square foot, 22, 2.2 square feet; 8 anodes of 8-mesh iron wire.

On October 5, 1900, I made an experiment with 22 sheet cathodes of 30-mesh wire cloth, and 8 anodes of 8-mesh wire cloth. The cathodes were $2\frac{1}{2}$ by $2\frac{1}{2}$ inches in size. The volume of the solution treated was 22 liters. The rate of flow was 60 liters per hour; the velocity of flow was 9.4 inches through the open box and 21 inches per minute through the cathodes. The estimated effective surface of one cathode was approximately 0.1 square foot, so that 22 cathodes

gave an effective surface of 2.2 square feet, and the actual surface of the wires was 2.52 square feet. The results of this experiment are shown in figure 6.

It will be noticed that there is a slow but continuous decrease in the value of the cyanide solution. The assay value of the silver begins at 16.91 mg. per 100 c. c., or 4.93 ounces per ton. The voltage is between 1.7 and 1.9 volts. Attention is called to the silver precipitation curve.

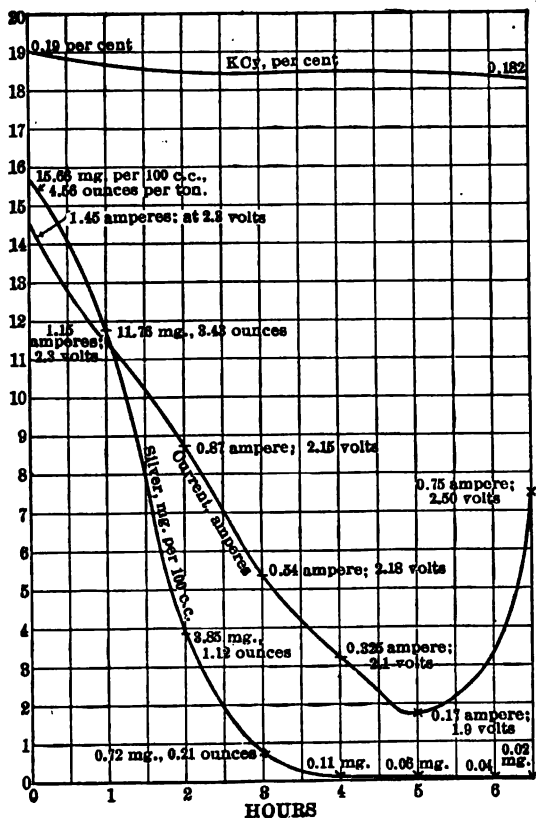


FIGURE 7.—Electrodeposition results with silver solution with rate of flow of 120 liters per hour; velocity, about 19 inches per minute through open box, 4½ inches through cathodes; anodes and cathodes same as in figure 6. Voltage varied.

experiment. The voltage in this experiment varied between 2.1 and 2.5 volts. In spite of the high current density, there was no gas visible on either cathodes or anodes, so long as the solution was in circulation.

Figure 8 shows the results of an experiment similar to that represented in figure 7, except that the area of cathodes was increased. Twenty-two old cathodes of 30-mesh iron wire and two new cathodes, each consisting of 10 sheets of 1-mm. wire cloth, were used. The effec-

tion of the fall of the silver is very rapid during the first three hours, during the next two hours it becomes slower, and after five hours it appears as an asymptote to the x axis, when the current is seen to be not economically used.

The effect of the increased rates of flow is shown in figure 7.

In the experiment here represented the rate of flow was increased to 120 liters per hour (velocity, 42 inches per minute through cathodes), or double that employed in obtaining the results shown in figure 6; other conditions were the same as in figure 6. It will be noticed that the silver was much more rapidly precipitated than in the previous

tive surface area of the cathodes was approximately 4.2 square feet, or twice that of the cathodes used for the previous test. In this experiment the voltage was controlled better than before. In the beginning 2.03 volts was employed, but after two and one-quarter hours the potential was raised to 2.5 volts, remaining thus until five and one-quarter hours had passed. At the end of the five and one-quarter hours a

new charge of KAgCy_2 was added, bringing the total volume again to 21.65 liters and replacing the solution used in the samples. It will be noticed that the current immediately rose from 0.6 ampere, which had been maintained by the 2.5 volts, to 1.45 amperes, and that the voltage fell to 2.35, showing that the solution had become a better conductor. It will also be noticed that the precipitation of silver was much better after a new charge had been added than before. Two hours after the silver solution had been added, the silver content had been

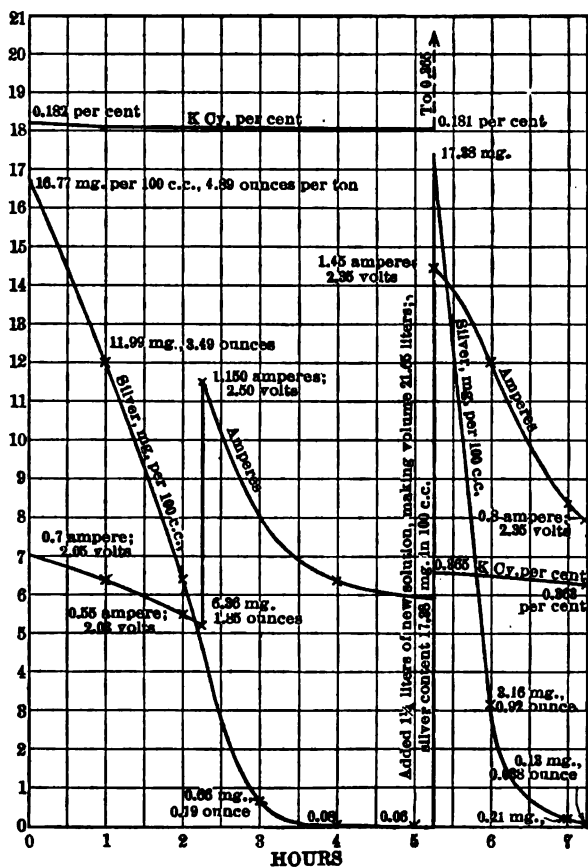


FIGURE 8.—Electrodeposition results with silver solution with rate of flow of 120 liters per hour and larger cathode area. Velocity of flow, 19 inches per minute through open box, 42 inches through cathodes; 22 used cathodes of 30-mesh wire cloth and 2 new cathodes, each of 10 sheets of 1-mm. wire cloth; actual wire surface of one cathode, — sq. cm.; estimated effective surface of 42 cathodes, 4.2 square feet.

reduced from 17.38 mg. per 100 c. c. to 0.13 mg., or 0.038 ounce per ton. As stated above, the area of cathodes exposed was twice the area of the cathodes shown in figure 6. The precipitation capacity of the box had increased enormously. It was possible to make a new precipitation after adding a new charge of silver at the end of two hours.

Figure 9 shows the results of an attempt to increase the flow of solution to 180 liters per hour. The whole of the solution passed through the box nearly nine times in one hour, and as the box held about a liter, an amount equal to 180 times the volume of the box passed through in each hour. Forty wire-cloth cathodes and seven wire-cloth anodes were used in this experiment. The cathodes were 3.5 by 2.6 inches square and had a total surface area of 6.38 square feet. It

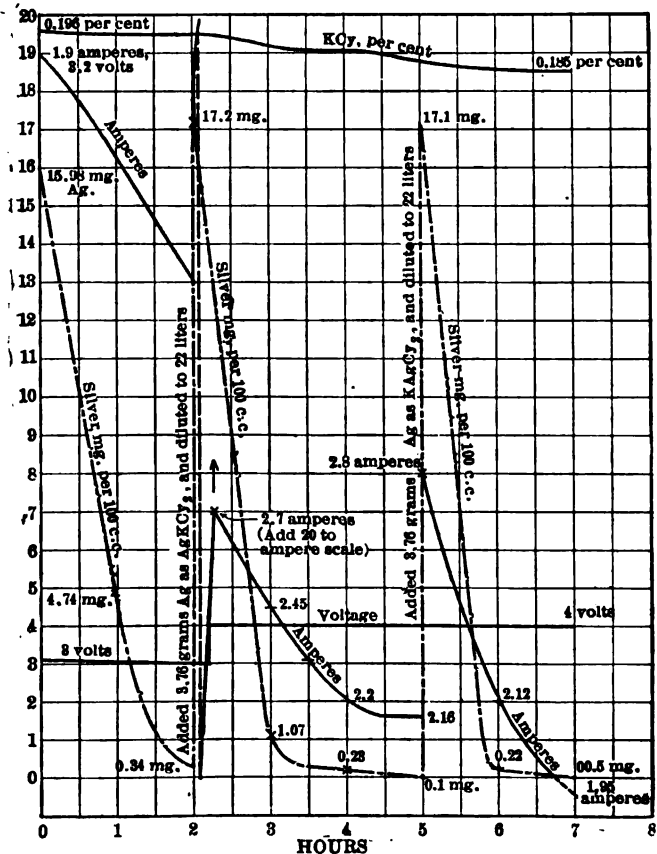


FIGURE 9.—Electrodeposition results with silver solution with rate of flow of 180 liters per hour; velocity, 20 inches per minute in open box, 45 inches through cathodes; 7 anodes of 8-mesh wire cloth; 40 cathodes of 30-mesh wire cloth; actual surface area, 6.38 square feet.

will be noticed that the precipitation curves show that the capacity of the box increased rapidly, a new charge being added at the end of two hours and another at the end of five hours. The voltage during the first two hours was 3, and during the rest of the time 4. The precipitation curves show successively better results.

As the precipitation became more effective, another feature noticed was that free cyanide was regenerated from the cyanide com-

bined with the precipitated silver. This feature was seen for the first time in the experiment represented in figure 10. In this experiment the flow of solution was increased to 300 liters per hour. The entire solution passed through the box 15 times in one hour. It will be

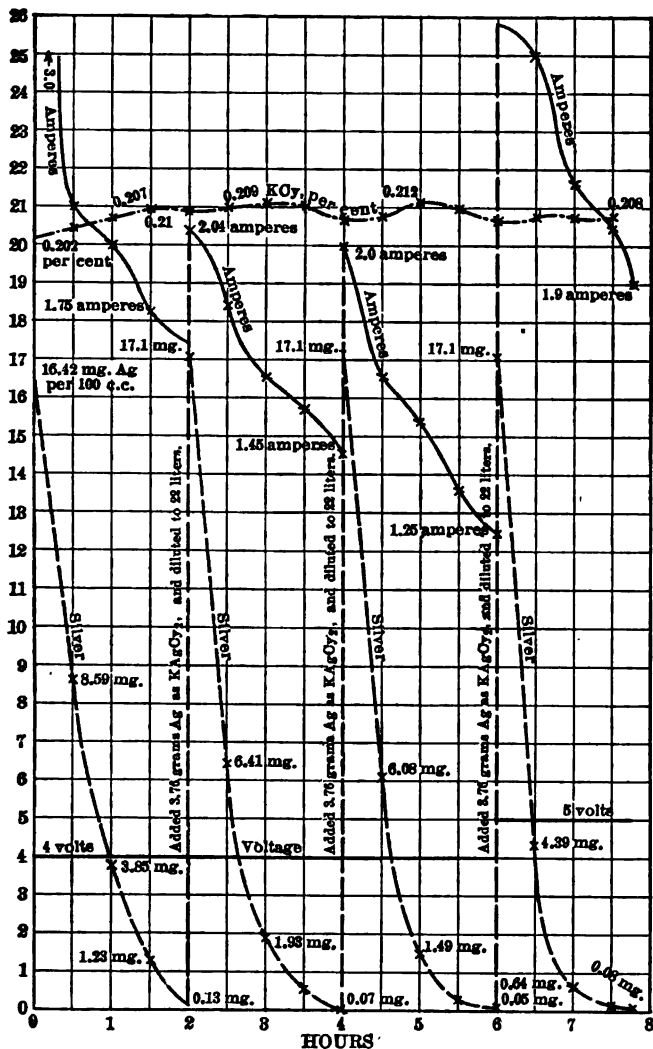


FIGURE 10.—Electrodeposition results with silver solution; rate of flow of 300 liters per hour; velocity through open box, 34 inches per minute; volume of solution, 22 liters; 7 anodes, 8-mesh wire cloth; 56 cathodes of 30-mesh wire cloth; wetted area, 3.5 by 2.6=9.1 square inches; actual surface area, 0.01773 square foot per square inch of cloth; for 56 cathodes, 8.936 square feet; length of electrodes, 5½ inches; volume, 47.7 cubic inches or 0.782 liter.

noticed that at the end of every two hours a new charge of $KAgCy_3$, with sufficient water to bring the volume up to 22 liters, was added

to the solution. The voltage was kept steady for the first six hours at 4, and during the last hour and a half was raised to 5 volts. It will be noticed also that the precipitation increased, especially between the sixth and seventh hours. The silver content and the current fell simultaneously, although the amperage did not fall quite as rapidly

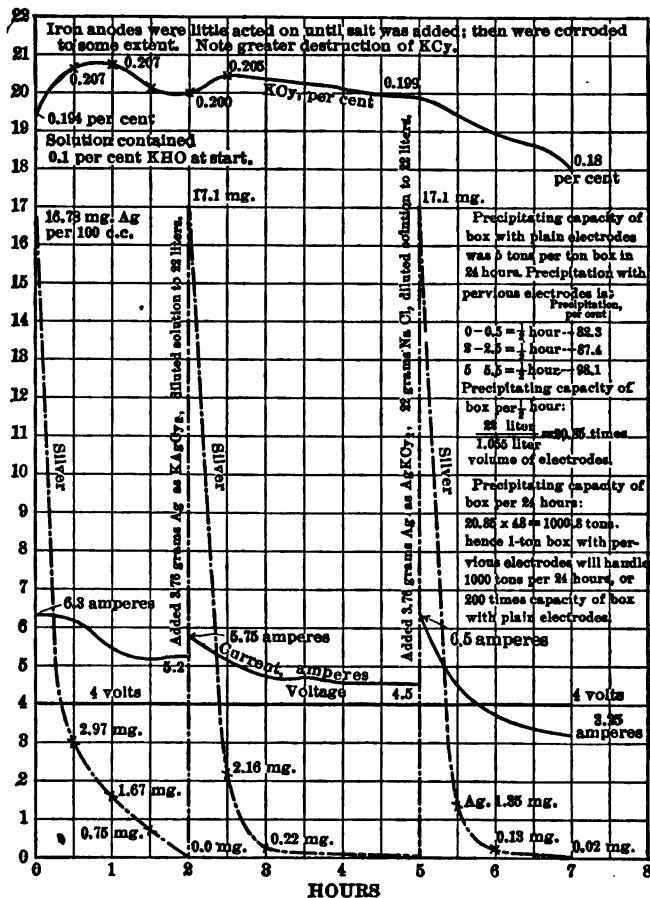


FIGURE 11.—Results of electrodeposition of silver with NaCl added to the solution; rate of flow, 300 liters per hour; velocity, 35 inches per minute through open box, 79 inches through cathodes; volume of solution, 22 liters; 11 anodes of 8-mesh wire cloth; 50 cathodes of 30-mesh cloth; total surface area, 5.735 square feet; length of electrodes, 18.75 cm.; volume occupied, 56.25 cm. or 1.055 liters.

as the silver content. It will be further noticed that during the rapid precipitation of the silver at the beginning of the experiment the solution contained 0.202 per cent cyanide, whereas at the end of one and one-half hours it had increased to 0.21 per cent. During the last half hour it fell again to 2.09 per cent. After a new charge of

KAgCy_2 , containing no free cyanide, had been added to replace the assay samples, the cyanide had again increased from 0.209 to 0.22 per cent, falling slightly again at the end of one and one-half hours, and rising each time a new charge of KAgCy_2 was added to the solution. There is no doubt about the titration of the cyanide.

The increase of cyanide during the precipitation period was seen still more clearly in the experiment represented in figure 11. The volume of solution and the rate of flow are the same as before. The number of cathodes is 50 and the voltage 4. Three charges of KAgCy_2 were treated, each containing 3.76 grams of silver. It will be noticed that during the first hour the cyanide had risen from 0.194 to 0.207 per cent, falling again in 30 minutes to 0.202 per cent as the silver was removed. When a new charge of silver cyanide was added it rose to 0.205, but fell at the end of three hours to 0.199 per cent. At the end of five hours a new charge of silver cyanide containing no free cyanide was added, together with 22 grams of salt. The salt was added to determine whether the increased conductivity of the solution, owing to the presence of the salt, would increase the rapidity of the precipitation. The salt did have somewhat such an effect. It will be observed, however, that there is a continued destruction of the cyanide by chlorine, resulting from the decomposition of the salt, which reduced the cyanide from 0.199 to 0.18 per cent.

CAPACITY OF BOX.

The estimated capacity of the box is shown in figure 11. The capacity of the box with plain electrodes without circulation, as already stated, was about 5 tons for a 1-ton box in 24 hours. In the first half hour, a precipitation of 82.3 per cent was obtained. In the first half hour of the second period a precipitation of 87.4 per cent was obtained, and in the first half hour of the third period the precipitation was 98.1 per cent. The volume treated was 22 liters, and the volume occupied by the 50 cathodes and 11 anodes was 1.055 liters. Dividing 22 by 1.055 we have for a half-hour period a precipitation capacity of 20.85 times the volume occupied by the electrodes, or 1000.8 for 24 hours. Clearly a box holding 1 ton, and fitted with electrodes as described, will handle over 1,000 tons in 24 hours. As a result of the investigation I was able to increase the precipitation capacity of the box filled with wire-cloth anodes and cathodes to more than 200 times that of a similar box with plain electrodes.

The effect of the increased capacity was shown still more strongly in the experiment represented in figure 12. The flow was increased to 720 liters per hour, and the voltage was increased to 9. The first treatment was for two hours, when 3.76 grams of silver was added to

replace the silver in assay samples. The recharging of the solution was repeated every hour for the next four hours, and the rapid regeneration of the cyanide is shown clearly by the upper curve. The rapid deposition of the silver is also shown, and the increased capacity of the box should also be mentioned.

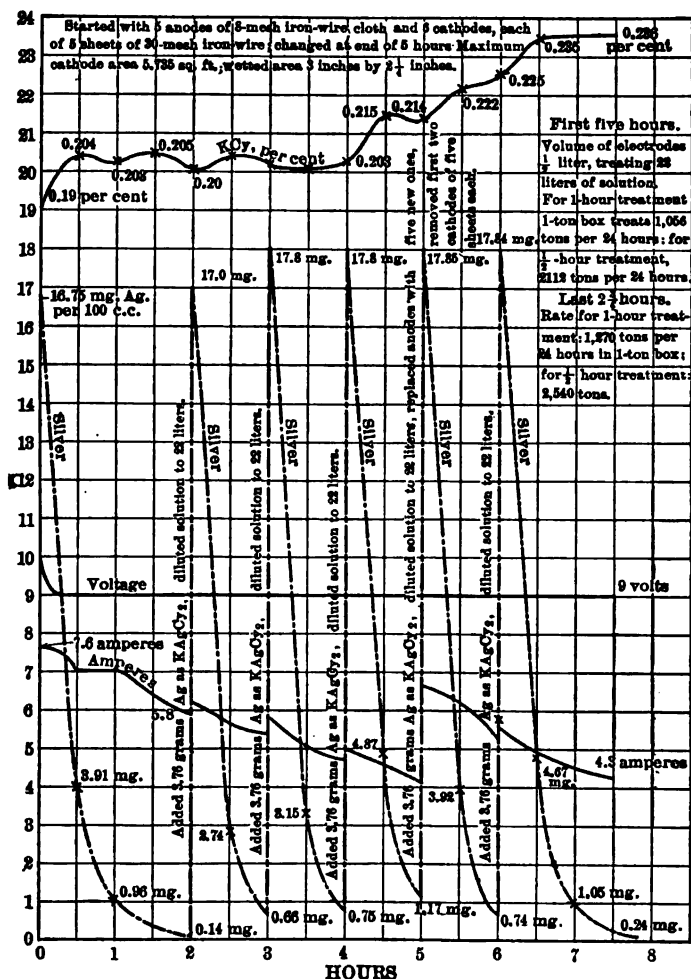


FIGURE 12.—Electrodeposition results with a rate of flow of 720 liters per hour; volume treated, 22 liters; velocity of flow, 84 inches per minute through open box, 189 inches through cathodes.

If the assay value of the solution be considered as reduced to about half an ounce of silver per ton, the capacity of the box can be figured at 1,056 tons per 24 hours for a 1-ton box; if the silver be considered to be reduced to a little more than an ounce per ton the capacity can be figured as 2,540 tons per 24 hours in a 1-ton box.

USE OF PEROXIDIZED-LEAD ANODES AND WIRE-CLOTH CATHODES.

In the preceding tests an important difficulty was encountered. The precipitation capacity of the box had been increased so rapidly that the 30-mesh wire cloth became clogged with precipitated silver, which interfered with the circulation of the solution. The wire-

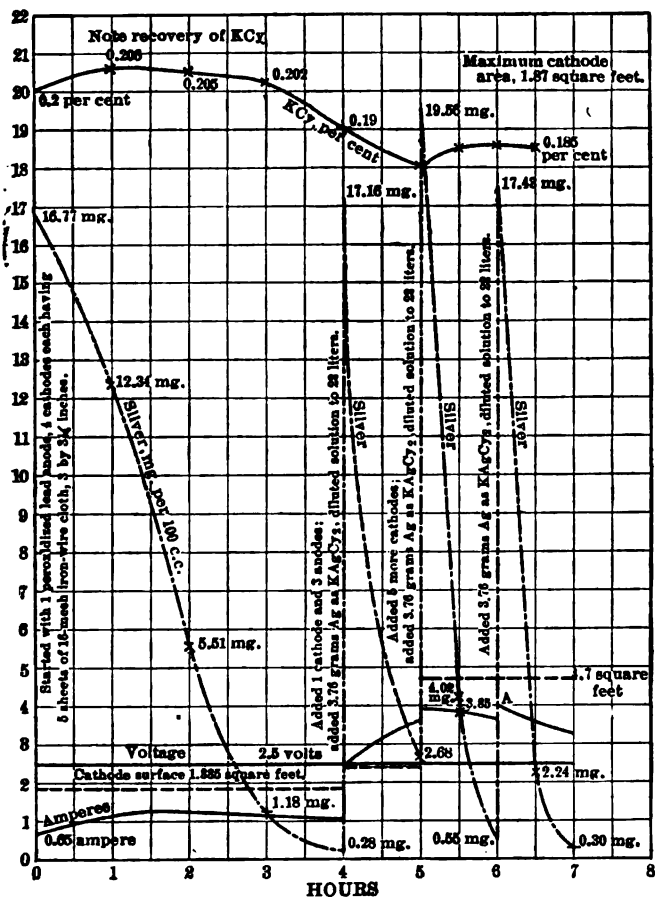


FIGURE 13.—Electrodeposition results with peroxidized-lead anodes and wire-cloth cathodes; volume of solution at start, 22 liters, containing 3.76 grams Ag as $KAgCy$, 22 grams KIO , and 45 grams KCy ; rate of flow, 420 liters per hour; velocity through open box, 63 inches per minute; through cathodes, 142 inches.

cloth anodes also had become somewhat coated with ferric hydrate. An attempt was made to obviate these troubles.

I started with one peroxidized-lead anode, made of perforated sheet lead, and with four cathodes each formed from five sheets of 16-mesh iron-wire cloth, 3 inches wide and $2\frac{1}{4}$ inches deep in the solution, a new box having been made for the purpose. The rate of flow was 7 liters per minute, or 420 liters per hour. With only one

ment. At first only 1 volt was applied, and instead of getting precipitation of silver, the silver already precipitated redissolved, as can be seen from the silver-precipitation curve in figure 15. The curve starts at 16.78 mg. per 100 c. c., and rises to 25 mg. per 100

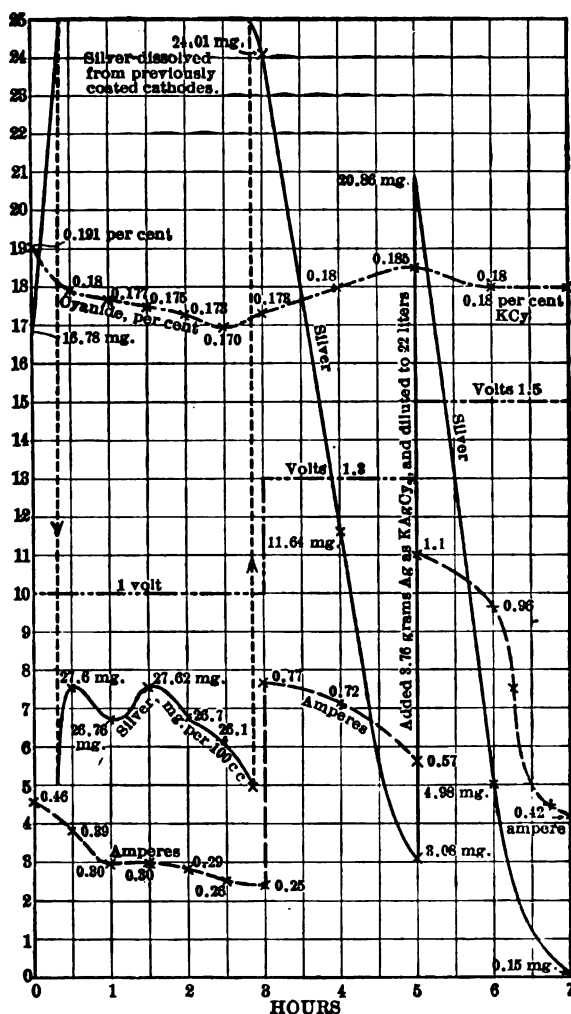


FIGURE 15.—Electrodeposition results with carbon anodes not coated with vaseline; 22 liters of water used, containing 44 grams KCy , 22 grams KHO , 3.76 grams Ag as KAgCy_2 ; rate of flow, 360 liters per hour; velocity through open box, 42 inches per minute; through cathodes, 96; 6 anode "combs" and 70 cathodes (same as in fig. 14) used; cathode area, 6.6 square feet. Owing to low voltage or poor contacts, some Ag was redissolved during first part of test and KCy absorbed.

c. c. in the first few minutes, then continues to rise and indicates 27.6 mg. per 100 c. c. at the end of the first quarter of an hour. The silver then decreased to 26.76 mg., increased again to 27.62, and fell again at the end of 2½ hours to 25. At the end of 3 hours the volt-

age was increased to 1.3 volts. At this point, the content of silver, which originally had been only 16.78 mg., had increased to 24 mg. With 1.3 volts, the silver began promptly to come down, and at the end of the next two hours the silver content had decreased to 3.08 mg. when a new charge of silver cyanide was added, bringing the content up to 20.86 mg. The voltage was increased to $1\frac{1}{2}$ volts, whereupon the silver was rapidly precipitated, being reduced in two hours to 0.15 mg., or about 0.05 ounce per ton. It will be noticed that during the whole time that silver had been dissolving from the cathode, free cyanide had been falling simultaneously. As soon as the silver began to be precipitated, it rose again, and at the end of the process, the cyanide content, which originally was 0.191 per cent, had fallen to 0.18 per cent. The experiment clearly shows that 1 volt is not sufficient for the effective precipitation of silver, that 1.3 volts is the minimum, and that a higher voltage than this is better.

EXPERIMENTS ON THE REGENERATION OF CYANIDE.

I have already called attention to the regeneration of the cyanide combined with the silver to be precipitated. The following experiments were undertaken to study the matter further. It was desired to avoid the oxidation of cyanide at the anode as much as possible; accordingly only $1\frac{1}{2}$ volts was employed. The solution used on a previous day was brought to a volume of 22 liters by adding the requisite amount of water. It then contained 0.189 per cent KCy and 0.1 per cent KHO. The rate of flow was 480 liters per hour. The electrodes occupied about 1 liter of space. The same anodes and cathodes were used as before (see fig. 14). The results of this test are shown in figure 16.

At the end of the first hour it is seen that the silver content had fallen from 17.36 mg. per 100 c. c. to 4.58 mg. leaving only about 1.74 ounces per ton in solution. Then a new charge of 3.76 grams of KAgCy_2 was added, and sufficient water to bring the volume back to 22 liters after the samples had been taken out. This brought the silver content up to 21.64 mg. per 100 c. c.

At the end of the second hour the silver content had fallen to 2.29 mg. per 100 c. c., or about 0.78 ounce per ton. A new charge of silver was added, as before, and at the end of the third hour the silver content had fallen to 0.74 mg. per 100 c. c., or 0.25 ounce per ton. A new charge was then added, and at the end of the fourth hour the silver had been reduced to 0.6 mg. per 100 c. c., or to 0.2 ounce per ton. A new charge of silver was then added, and at the end of the fifth hour the content had fallen again to 0.28 mg. per 100 c. c., or to 0.09 ounce per ton. A new charge was then added, and at the end

of the sixth hour the content had again fallen to 0.7 mg. per 100 c. c., or to 0.25 ounce per ton. A new charge was then added, and at the end of the seventh hour the content had been reduced to 0.33

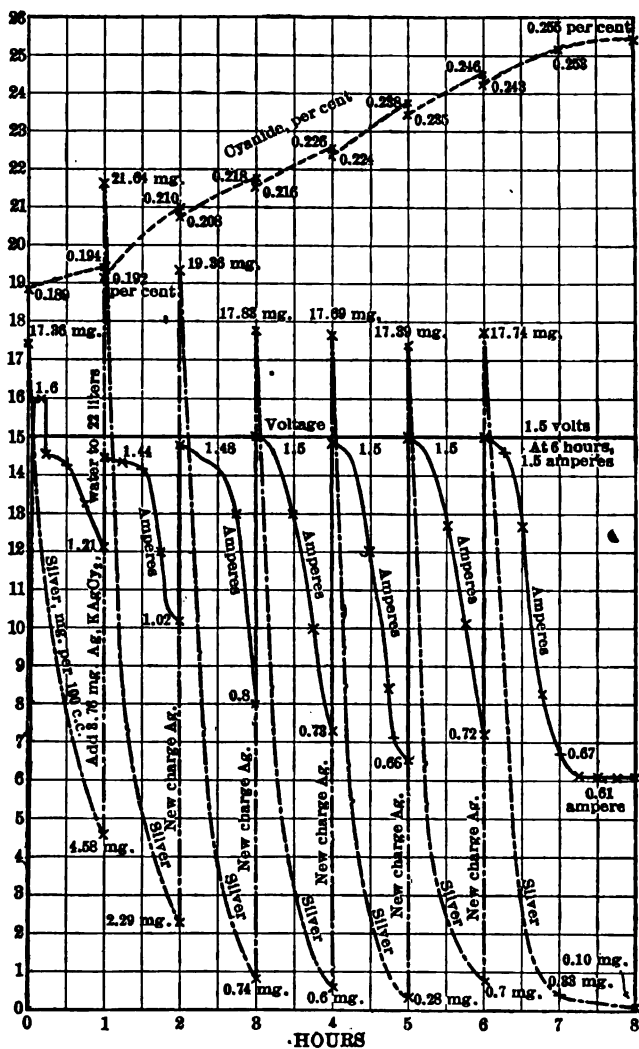


FIGURE 16.—Electrodeposition results showing regeneration of cyanide. Eight liters of water added to previously used silver cyanide solution to make up 22 liters, containing about 22 grams of KHO. Anodes and cathodes same as in fig. 15. Rate of flow, 480 liters per hour; velocity per minute through open box, 56 inches; through cathodes, 127. At 1, 2, 3, 4, 5, and 6 hours added 3.76 grams Ag as KAgCy and 0.25 liter of water to bring volume up to 22 liters. Note recovery of cyanide.

mg. per 100 c. c., or 0.11 ounce per ton. The test was continued for another hour, at the end of which time the silver in solution had been reduced to 0.1 mg. per 100 c. c. or 0.03 ounce per ton.

It will be observed that the cyanide titration shows a continuous rise in cyanide content. At the start the cyanide content was 0.189 per cent, and at the end of the first hour it had risen to 0.194 per cent, dropping again on account of dilution with water to 0.192 per cent. At the end of the second hour it had risen to 0.21, dropping again by dilution to 0.208. At the end of the third hour it had risen to 0.218, dropping again by dilution to 0.216. At the end of the fourth hour it had risen to 0.226, dropping again by dilution to 0.224. At the end of the fifth hour it had risen to 0.238. At the end of the sixth hour it had risen to 0.246, again dropping by dilution to 0.243. At the end of the seventh hour it had risen to 0.253, and at the end of the eighth hour it had risen to 0.255 per cent.

The total actual rise in the cyanide content, as shown by titration, was 0.066 per cent. If to this be added the amount of cyanide in the quantity of solution taken out for the assays, which was about 250 c. c. each time, the total saving of cyanide is 0.084 per cent. Multiplying this result by the weight of the solution, taken as 22 kg., gives an actual saving of 18.48 grams KCy, which was formerly combined with about 22.56 grams of silver. Taking the molecular weight of KCy as 130 and the atomic weight of silver as 108, if all the cyanide had been recovered there would be 1.2036 grams of KCy for every gram of silver precipitated. On this basis, as there was, roughly, 22.56 grams silver precipitated, there would be saved 27.07 grams KCy. The amount actually recovered was 18.48 grams, or slightly more than 68 per cent of the cyanide originally combined with the silver. It is evident, therefore, that all of this cyanide was not recovered, but a sufficient amount was saved to make this saving a highly important factor in the cost of precipitation.

It will be noticed that the curves representing the cyanide content all are rising where the silver is being rapidly precipitated. Between the sixth and seventh hour, and particularly at the end of the seventh hour, the curve of the recovered cyanide rises more rapidly than later, when the silver has been mostly precipitated. Toward the end of the experiment the destruction of cyanide begins to prevail over the regeneration, whereupon the curve rises more slowly, and finally begins to fall. This is caused by oxidation, which is constantly taking place at the anodes.

Attention is particularly directed to the parallelism between the fall of the ampere curves and those showing the precipitation of the silver. This is most clear between the sixth and eighth hours, where the ampere curve falls abruptly from 1.5 amperes. When the silver has fallen to 0.2 mg. per 100 c. c., and has been practically exhausted, as at the end of $7\frac{1}{4}$ hours, the ampere curve has fallen to 0.61. Here it remains constant during the rest of the experiment.

This phenomenon suggests a very simple method of controlling the precipitation without assaying the solution. If the voltage is maintained constant, the ampere curve shows the critical point at which it is best to stop precipitation. The point at which the ampere curve becomes constant is the most economical point to stop precipitation. At this point the recovery of cyanide reaches the maximum, and a further continuation of precipitation is wasteful of electric current.

During this test assay samples were taken every 15 minutes, but in practice sampling at longer intervals would suffice to correct the reading of the ammeter. This method of controlling the work I consider to be of great practical importance industrially, and the benefit can be gained only where a large volume of solution which is being rapidly circulated is treated, as here shown. The subject of cyanide regeneration is considered more fully in subsequent pages.

RUSTING OF IRON-WIRE CATHODES.

Attention should be called to another point that is evident in nearly all the later tests, but is also shown very clearly in the foregoing experiment. As the experiments were conducted intermittently and could only be carried on in the daytime, between the tests the wire-cloth electrodes became more or less rusted by the action of moisture on the wires, which were coated with silver. As is well known, under such circumstances a local action sets in, which causes rusting of the iron wire. When the cathodes were first immersed in the solution, much of the electric current was consumed in reducing this oxide of iron, and hence was not available for the precipitation of silver. It will be noticed that the precipitation improves during the continuation of the process. Other causes of this increase, aside from the dissolution of the rust, are that the conductivity of the solution gradually increases, owing to the presence of a larger amount of cyanide and other salts, and that the surface of the wire-cloth cathodes becomes covered with silver, making them better electrical conductors. From these three reasons it is evident that the rusting of the wire cloth will not take place whenever it is possible to have the process in continuous use.

TREATMENT OF RICH SILVER CYANIDE SOLUTIONS.

EXPERIMENT WITH SODIUM CHLORIDE IN THE SOLUTION.

The experiments hitherto described were conducted with silver solutions that were comparable in richness with those that occur in the treatment of ordinary ores. Experiments were next undertaken with a richer solution, containing 119.11 mg. of silver per 100 c. c. (see fig. 17), or about 41 ounces per ton. In other respects the solu-

tion treated was similar to those used before, consisting of 22 liters of 0.1 per cent free KHO, with 0.237 per cent KCy. In order to increase the conductivity of the solution, about 1 per cent of NaCl was added, and the circulation was at the rate of $9\frac{1}{2}$ liters per minute, or 570 liters per hour. Six uncoated carbon anode combs were used; also 14 "bunches of five," or 70 in all, of the 16-mesh iron-wire cathodes, with a total area of 6.6 square feet. The voltage was 1.5.

Regeneration of the cyanide reached a maximum in this experiment at the end of two and one-half hours, when the cyanide content had risen to 0.295 per cent, after which it slowly declined to 0.273 per cent, owing to the action of chlorine and oxygen, set free at the anode. It will be noticed that there is again a close parallelism between the amperage curve and that of the rate of precipitation of the silver. Both curves change direction rapidly a little after three hours, the amperage falling from 2.57 to 0.70 at the end of that time, and the silver from 119.11 to 2 mg.

The effect of stopping circulation is shown at the end of three and one-fourth hours, at which time the belt on the pump accidentally broke. The current immediately

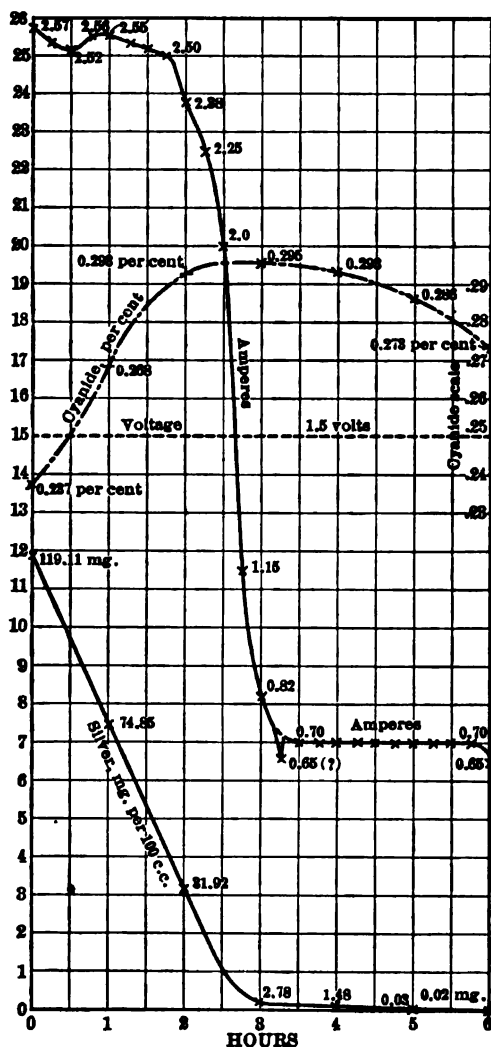


FIGURE 17.—Electrodeposition results with richer solution. Twenty-two liters of solution, containing 119.11 mg. of silver, 22 grams KHO, 220 grams NaCl, and 0.237 per cent KCy. Six uncoated "Electra" carbon anode combs. Cathodes, 14 bunches of 5 each, of 16-mesh wire cloth; total area, 6.59 square feet. Rate of flow, 570 liters per hour; velocity through open box, 66 inches per minute; through cathodes, 150.

dropped from 0.73 to 0.65, rising again to its place on the curve as

soon as the pump was started. At the end of five and three-fourths hours, the circulation being maintained, the current was 0.70, and when the circulation was stopped it fell at once to 0.65.

EXPERIMENT WITHOUT SODIUM CHLORIDE.

In order to determine whether the use of salt in the solution was beneficial the experiment was repeated without salt. The same vol-

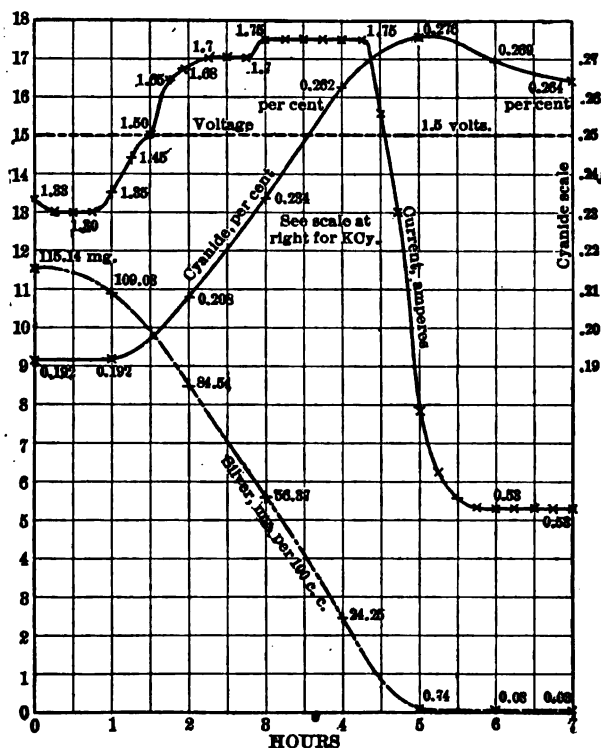


FIGURE 18.—Electrodeposition results with rich silver solution and no NaCl. Solution, 22 liters, containing 42.24 grams KCy, 25.33 grams Ag as KAgCy, and 22 grams KHO. Rate of flow, 540 liters per hour; velocity per minute through open box, 63 inches; through cathodes, 141. Anodes and cathodes same as in figure 17; cathodes were coated with silver and rusted from standing over night after removal from the salt solution used in the previous test.

ume of solution, containing 0.1 per cent KHO and 0.192 per cent KCy and 115.14 mg. of silver per 100 c. c. was used. The flow was 540 liters per hour. The same cathodes, which had been coated with silver, were used as in the previous experiment (fig. 17). The results of the test are shown in figure 18. The ampere curve was decidedly different in form from that of the former experiment. It started at 1.33 amperes, falling in one-fourth hour to 1.30, then rising rapidly to 1.75 at the end of three hours. This value it main-

tained until four hours and twenty minutes of the total period had passed, when it fell precipitately until, at the end of five and three-fourths hours, it had become 0.53 ampere. There it remained constant until a total of seven hours had passed, when the experiment was stopped. It will be noted that the increase of cyanide content did not begin until one hour had passed, when it rose from 0.192 to 0.276 per cent. At this point most of the silver had been precipitated. At the end of five and one-half hours the cyanide content began to fall, and at the end of the test was 0.264 per cent.

It is evident from a study of these curves that the experiment should have been stopped at the end of five hours, when the recovery

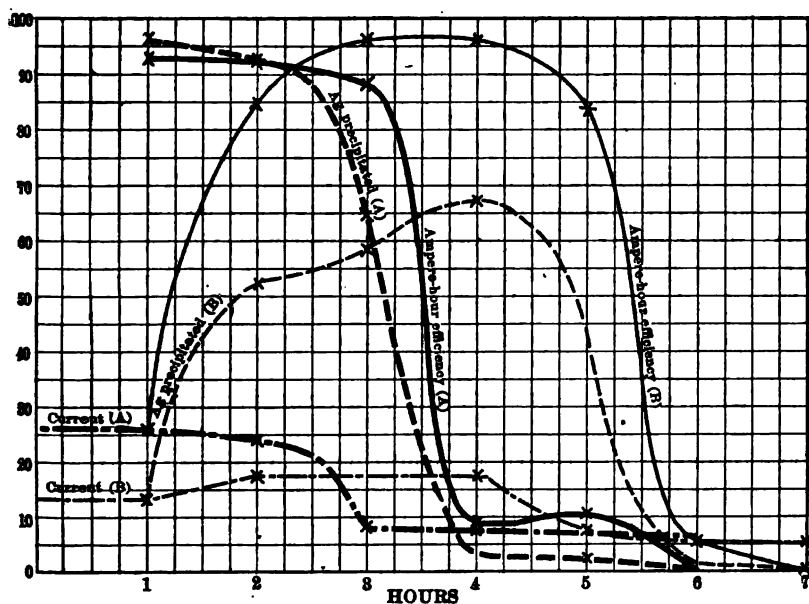


FIGURE 19.—Results of tests with and without use of salt compared. A, curves from test shown in fig. 17, salt used. B, curves from test shown in fig. 18, no salt used.

of cyanide was at the maximum. After that time the current was not used economically, and little silver was precipitated. The irregular rise of the current was evidently due to the rusting of the cathodes, which had stood over night wet with salt solution. The silver did not begin to come down rapidly, and the cyanide to be regenerated, until this rust had been reduced by the current. If the cathodes had been perfectly free from rust no doubt the current would have started at about 1.75 amperes, and would have remained nearly constant, and the silver would have been precipitated much earlier in the experiment. The voltage and amperage were read every 15 minutes with Weston standard meters. At the start and at the end of each hour a 250-c. c. sample was taken for cyanide titrations and for silver

assays. It will be observed that the electrical efficiency was extremely high as long as there was any silver to be precipitated, but that it fell rapidly when the silver content became diminished. The relative efficiency in the two tests is best shown in figure 19.

COMPARISON OF RESULTS.

If, neglecting the oxidation at the anode, we assume that the silver is precipitated at the cathode by the potassium ion, then for every atom of silver precipitated there will be two molecules of KCN set free at the cathode. Assuming 65 as the molecular weight of KCN, for every 108 grams of precipitated silver 130 grams of KCN should be regenerated, or $\frac{130}{108}=1.2036$ grams of KCN regenerated for each gram of silver precipitated. Evidently, notwithstanding the initial resistance to the current due to the rusting of the anodes, the final efficiency of both the silver precipitation and the cyanide regeneration was better in the second experiment, shown in figure 18, than in the first, shown in figure 17. In the first experiment the highest silver efficiency was 93.62 per cent, whereas in the second it was 96.35. In the first the highest cyanide regeneration was 58.19 per cent, but in the second it rose to 76.67 per cent. Evidently the addition of sodium chloride is not justifiable. The irregularity of the curves in the second experiment was also due to this cause.

THEORY OF ELECTROLYTIC CYANIDE REGENERATION.

The theory of electrolytic regeneration of cyanide is of great practical importance and is difficult to explain in a few words. According to the modern view the ions that carry the current in the solution exist already dissociated, or, at most, loosely associated. The electric current, accordingly, has only to direct their motion. The anode, being the positive pole of the cell, attracts the anions, each of which carries a negative charge to the anode where it gives up its negative charge. Simultaneously the cathode, being the negative pole of the cell, attracts the cations, each of which carries a positive charge and delivers it to the cathode. By this means each of the ions travels through the solution a variable, relative distance dependent on the "transfer velocities." The velocities have been accurately measured by Hittorf and others for many electrolytes. They are not necessarily equal; often they are relatively quite different; but for simplicity let us assume that all are equal. The potassium cation and the chlorine anion in normal solutions of KCl have nearly equal values.

Let us first assume a solution containing ions arranged as shown in the following diagram:

	+		+		+		+		+		+	
	-		-		-		-		-		-	
1	2	3	4	5	6	7	8	9	10	11	12	13

In this arrangement the + sign represents, for example, the potassium ion with its positive charge, the - sign represents the chlorine ion with its negative charge. If into such a solution are inserted two platinum electrodes with + and - charges the equilibrium is disturbed and the anions with their negative charges move one place to the right toward the anode with its negative charge, and each cation moves one place to the left with its positive charge, toward the cathode with its negative charge. The arrangement then becomes as follows:

-		+		+		+		+		+		+
		-		-		-		-		-		-
1	2	3	4	5	6	7	8	9	10	11	12	13

When the chlorine anion, with its negative charge, reaches the anode, it gives up this electrical charge and becomes gaseous chlorine which escapes. When the potassium cation with its positive charge reaches the cathode it gives up its positive charge and becomes metallic potassium, which by a secondary reaction decomposes the adjacent water and produces KHO which remains in solution and hydrogen gas which escapes as such. It will be noticed that the intervening solution remains unchanged in chemical composition and that the only chemical changes occur at the electrodes. The theory thus agrees with the facts first observed by Faraday.

As regards potassium silver cyanide, the potassium cation with its positive charge travels a little faster than the AgCy_2 anion with its negative charge. The difference of velocity is not great and the preceding diagram will suffice to explain the case. We have only to suppose the cation with the + sign to be potassium as before and the anion with the - sign to be not chlorine but AgCy_2 . There will then be an increase of one molecule of AgCy_2 at the anode and an increase of one atom of potassium at the cathode. Suppose that there also are evenly diffused through the solution other molecules of KCy , KHO , KAgCy_2 , not yet brought under the influence of the current. Then when a K cation reaches the cathode and gives up its positive charge

it finds KAgCy_2 there, which is more easily decomposed than water. A secondary action, shown in the equation following, is then supposed to ensue.



This reaction was first suggested by Hittorf to explain the smooth, silver deposit obtained from cyanides, in contrast with the loose crystalline deposit from other salts of silver, such as the nitrate, where the silver is directly deposited by the current.

It will be noted that after electrolysis two atoms of silver are removed from the solution at the cathode, one by electrolytic wandering to the anode, the other by the precipitating action of the cation. (As regards the nitrate there would be removed only the one atom directly precipitated in crystalline form by the electric current.)

Let us next suppose this process to be repeated at the cathode, and further, that throughout the solution, not yet affected by electrolysis, there is uniformly diffused 2 KCy in the free state, 2 KHO , and 2 KAgCy_2 . At the cathode there is, then, before electrolysis: $2\text{KCy} + 2\text{KHO} + 2\text{KAgCy}_2$, and after electrolysis (as 2 K ions appear): $2\text{KCy} + 2\text{KHO} + 2\text{KAgCy}_2 + 2\text{K} - 6\text{KCy} + 2\text{Ag} + 2\text{KHO}$. It is seen that there is a gain at the cathode of 4 KCy , free, for each 2 Ag precipitated.

Let us make a similar supposition concerning the solution at the anode. Before electrolysis the composition was, $2\text{KCy} + 2\text{KHO} + 2\text{KAgCy}_2$, and after two units of electricity have passed, it is $2\text{KCy} + 2\text{KHO} + 2\text{KAgCy}_2 + 2\text{AgCy}_2$, an increase of two molecules of AgCy_2 at the anode. Now as shown by Morgan, 2AgCy_2 easily splits up into $2\text{AgCy} + 2\text{Cy}$.^a

The 2AgCy easily combines with 2KCy to form 2KAgCy_2 , and the 2Cy reacts with 2KHO to form H_2O , KCyO , and KCy , one half being destroyed by becoming cyanate, the other half being regenerated as KCy . Combining these reactions at the anode into one we have, after electrolysis, the following chemical substances: $4\text{KAgCy}_2 + \text{KCyO} + \text{KCy} + \text{H}_2\text{O}$. Hence at the anode there is a gain of two molecules of KAgCy_2 , one of KCyO , one of water, a loss of two molecules of KHO and two of KCy , and a gain of one molecule of KCy ; therefore there is a net loss at the anode of KCy . But there was a gain at the cathode of 4 KCy , hence the net result is a total gain of 3 KCy for each 2 Ag precipitated, and the weights of silver precipitated and cyanide recovered will be:

$$\frac{\text{Cyanide recovered}}{\text{Silver precipitated}} = \frac{3 \times 65}{2 \times 108} = \frac{195}{216} = 90.27 \text{ per cent.}$$

^a Christy, S. B., Electromotive force of metals in cyanide solutions: Trans. Am. Inst. Min. Eng., vol. 30, 1900, p. 882.

If no cyanide were oxidized, for every part of silver precipitated there would be: $\frac{2 \times 65}{108} \times 100 = \frac{130}{108} \times 100 = 120.37$ per cent cyanide recovered. As will be seen later, by greatly increasing the cathode area as compared with the anode area, I have actually recovered free cyanide to the extent of 118 per cent of the weight of the silver precipitated.

The reactions with potassium auro-cyanide are exactly the same as with those of silver, AuCy_2 being substituted for AgCy_2 . (It is possible that auric-cyanides are formed at the anode to be afterwards reduced to aurous at the cathode.) As the atomic weight of gold is greater than that of silver, however, the relative weights of cyanide recovered from a given weight of gold will be $\frac{108}{197} = 0.55$ times the amount recovered by an equal weight of silver. Further, as gold solutions usually contain very little gold cyanide compared with those obtained in treating rich silver ores, the amount regenerated is seldom noticeable.

Theoretically, cuprous cyanide should permit a large regeneration of cyanide, owing to its low atomic weight. I have been unable to obtain certain proof of this regeneration in the few experiments I have tried. Mr. Hamilton, in experiments with the Butters process, claims to have obtained such evidence. I shall discuss this later. The facility with which both gold and copper form complex cyanides of higher valency at the anodes by taking on one or more Cy radicals possibly explains the greater difficulty of regenerating cyanide from them than from silver cyanides. With regard to possible regeneration of cyanide from potassium cyanate and sulpho-cyanate, as claimed by Clancy, I will say that many years ago I made some preliminary experiments on this subject and got some small traces of regeneration, but not enough to cause me to follow the matter further. The regeneration of cyanide up to 118 per cent of the weight of silver precipitated as previously described would seem to indicate that some of the cyanate must have been regenerated.

Evidently, there is the certain possibility of regenerating a large part of the cyanide combined with silver, particularly in the rich solutions. This gives to electrolytic methods a great advantage over the usual methods of zinc precipitation. But to take full advantage of such regeneration the silver precipitation must not be pushed too far, for when the silver is nearly gone, electric current is wasted in decomposing the water, and the nascent oxygen set free at the anode destroys more cyanide than is regenerated at the cathode, particularly when the surface of the anode equals or exceeds that of the cathode. Of course actual mill solutions contain many other substances than those I have supposed, and the reactions are much more complicated

than those I have outlined. The final result depends not only on the composition of the solution, but also on the voltage, the relative current density at anode and cathode, and even on the physical as well as the chemical nature of the cathodes themselves. The tendency of cyanogen molecules to combine with one another, forming paracyanogen, and to oxidize to azulmic acid is another complication, as is the tendency, which should be mentioned, of cyanide to deteriorate into K_2CO_3 and ammonia compounds. The presence of sufficient free alkali is of controlling importance in the regeneration of the cyanide.

CAN CYANIDE BE REGENERATED BY THE ELECTROLYSIS OF FERROCYANIDES?

Prof. Kern,^a of Columbia, in a report published in 1913, answers this question with an emphatic "No!" He says:

(12) The regeneration of cyanide solutions which contain sulphocyanide and ferrocyanide does not occur by electrolysis by direct current, whether the conditions of electrolysis be made oxidizing or reducing by varying the relative current densities at the anode and the cathode.

On March 14, 1901, I made a preliminary investigation to see whether there was anything in this idea. Twelve liters of a 1 per cent solution of K_4FeCy_6 , with 0.2 per cent KHO, was made up. It was circulated through the deposition box. Six hard carbon comb anodes were used and 14 of the 5-cluster clean iron-wire cathodes.

The original solution was carefully titrated for KCy and appeared to contain a trace (0.001 per cent) of KCy, which possibly was produced by the action of the potash. The solution was clear and of a pale yellow color. It was electrolyzed for four hours at 1.5 volts, and 1.4 to 2.1 amperes. The solution changed in color, taking on a slightly greenish tinge, and traces of a deposit like Fe_2O_3 formed on the carbon anodes. Seemingly there was a very slight increase in the reaction for cyanide, but this was doubtful.

The carbon anodes were then removed and platinum was substituted for them. The voltage was increased to 2.5, and the current of 3.3 to 4.2 amperes was continued for one hour. No definite change was observed in the cyanide. A little ferric hydrate and Prussian blue formed on the anodes.

The voltage was then raised to 4, and the current increased to 8 amperes and was continued for one hour. The solution was now full of fine bubbles and had turned orange color, due probably to suspended ferric hydrate. A distinct reaction for cyanide was obtained, the titration showing 0.005 per cent KCy. The anodes were coated with ferric hydrate and Prussian blue. In 20 minutes more

^a Kern, E. F., *Electrolysis of cyanide solutions*: Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 265.

the KCy content rose to 0.007 per cent, and one hour later to 0.019 per cent. The entire solution was full of minute bubbles of gas, the composition of which was not determined. Also, loose flocculent ferric hydrate was contained in suspension. Of the original sample, which titrated 0.001 per cent KCy before electrolysis, 22 c. c. was inclosed in a tube with 12 c. c. of air and with a standard gold strip 2 inches by one-fourth inch in size, weighing 0.25 gram. The gold strip was then rotated for one hour in the solution, after which time it was found to have lost in weight 0.25 mg. gold. The solution therefore must have contained a trace of cyanide produced by the dissociation of the ferrocyanide in the presence of the KHO.

A similar experiment was made with 22 c. c. of the clear solution (titrating 0.019 per cent KCy), taken at the end of the electrolysis, and 12 c. c. of air. Both tubes containing the gold were rotated on the same shaft for one hour side by side. Upon weighing the gold, the loss from this second strip was 4.05 mg., or more than 16 times that from the first strip.

The gold strips were then reversed, the second being placed in the original solution and the first in the residual solution. The former now lost 0.04 mg. and the latter 0.94 mg. A film, not visible to the eye, seemed to have formed on both gold strips, but the electrolyzed solution was still leading, being this time 23.5 times as effective as the original solution. It would appear from these experiments that with platinum electrodes, at least, cyanide can be regenerated from potassium ferrocyanide in the presence of caustic potash, with a current of moderate density. This would give a great advantage to electrolysis if it could be made generally applicable. However, I did not follow the matter further.

Although Kern denies the possibility of any regeneration from the ferrocyanide by direct current, he says:*

When cyanide solutions containing sulpho or ferrocyanide were electrolyzed, the cyanide consumption was much less than that which occurred in pure cyanide solutions, which indicates that sulphocyanide and ferrocyanide act as protective agents during the electrolysis of cyanide solutions.

It would seem that another explanation than that of "protective agents" is possible—namely, that there was actually a slight regeneration that partly offset the usual oxidation losses. The amount of cyanide that was regenerated in my test was slight. It was necessary to use 100 c. c. samples to be sure of the cyanide titration. However, there seems no doubt that under certain conditions a small amount of cyanide can be regenerated from the ferrocyanide by the direct current, and this factor is another possible advantage in electrolysis over zinc precipitation.

* Kern, E. F., work cited, p. 264.

SHEET-IRON ELECTRODES COMPARED WITH IRON WIRE-CLOTH ELECTRODES.

To determine whether the iron wire-cloth electrodes were actually superior to those of sheet iron, 12 sheet-iron anodes were made. These sheets, $3\frac{1}{2}$ inches high by 3 inches wide, were held by two hard rubber strips, one on each side, so extended as to keep the lower edge $\frac{1}{4}$ inch above the bottom of the box. The 12 sheet-iron cathodes were 4 inches high by 3 inches wide, in the clear, and were held in ebonite insulating strips. Four $\frac{1}{4}$ -inch perforations were made near the top of each strip to allow the solution to circulate. The cathodes were coated with a thin film of graphite and vaseline. The electrodes were $\frac{1}{4}$ inch apart. The effective cathode area was 1.25 square feet or about 0.12 square meter. Circulation was produced by the centrifugal pump, and the flow was up and down through the $\frac{1}{4}$ -inch by 3-inch spaces, at the rate of only 0.56 liter per minute in the beginning. The results were as follows:

Results of tests with sheet-iron electrodes.

Time, hours.	Volts.	Amperes.	KCy in solution, per cent.	Silver per 100 c. c., mg.
0	0	0	0.215	16.75
1	1.5	0.60-0.22	.214	13.99
2	1.5	.27	.214	12.06
3	1.5	.265	.213	10.43
4	1.5	.23	.212	9.13
5	1.5	.24	.210	8.65
6	1.5	.21	.209	8.25
7	1.5	.20	.209	6.86

The results (see fig. 20) at the end of the first hour were so poor that during the second hour the cathodes were removed, cleaned with gasoline of the graphite-vaseline mixture and of silver, and then replaced. At the end of the fourth hour the electrodes were placed parallel with the longer dimension of the box to increase the velocity of flow to 6 liters per minute. The experiment was repeated with many variations without graphite and vaseline, in which a voltage of 2 volts and a current of 1 ampere were employed, and also 2.5 volts and 3 amperes. With the stronger current the precipitation was very little better. At 0.95 ampere small bubbles of gas filled the solution and this gas increased with higher current densities. The advantage of the greater surface presented by the wire-cloth cathodes compared with that of the sheet-iron cathodes was evident.

COMPARISON OF SILVER CLEAN-UP FROM SHEET-IRON ANODE WITH CLEAN-UP FROM WIRE-GAUZE ANODES.

The clean-up box that was used held 500 c. c. of solution; the solution contained 0.1 per cent KCy. The voltage was not recorded;

from memory I judge that it was derived from one Daniell cell and was higher at the end than in the beginning. The anode was of sheet iron 8.3 by 8.7 cm. in size, giving an area of 144.2 square centimeters for the two sides. The current was 0.1 ampere at first, finally becoming 0.005 ampere. The sheets of silver that finally

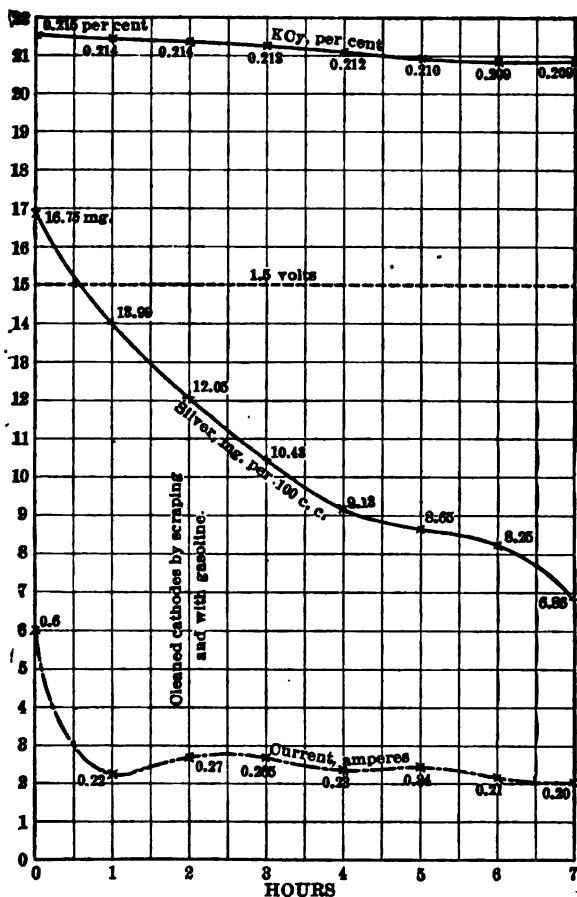


FIGURE 20.—Electrodeposition results with sheet-iron electrodes. Twelve anodes, effective area 1.25 square feet, and 12 cathodes, effective area 1.25 square feet, spaced $\frac{1}{4}$ inch. Silver cyanide solution, rate of flow 0.56 to 6 liters per minute. During first four hours, electrodes vertical, velocity of flow up and down slots averaged 0.68 inch per minute. After fourth hour, electrodes parallel with flow, velocity much increased. Figure 16 (p. 66), compared with this figure, shows advantage of using wire-cloth cathodes and larger area of cathode than of anode.

were peeled off were about 0.23 mm. thick; one weighed 19.78 grams and the other 16.85 grams. To the total weight of the sheet silver recovered, 36.63 grams, should be added that of the silver slimes from solution (0.39 gram), making a total of 37.02 grams. The residual cyanide solution, which was a deep yellow, contained

0.065 per cent free cyanide and 0.153 gram iron per liter. Half of this weight of iron had dissolved from the anode.

Another clean-up was made during which the stripping solution was made to circulate through the clean-up box. The cathode used was $2\frac{1}{4}$ by $3\frac{1}{2}$ inches in size, giving an area for the two sides of 19.26 square inches or 0.134 square foot. With 10 wire-cloth wire cathodes in two bunches of five, connected as anodes on each side of the cathode, 1 Daniell cell gave 0.25 ampere. When the solution was strongly circulating, the current rose to 0.35; on the circulation being stopped, the amperage fell in one minute to 0.1 and the voltage rose to 0.7. Finally the current fell to 0.2 ampere, and the cyanide content to 0.25 per cent; and the solution contained 29.73 mg. of silver per 100 c. c. The solution was now strengthened to 0.45 per cent free KCy and the voltage was raised to 1.25, when the last two bunches of five cathodes from the deposition box were put in.

The current density was at first 2.61 and finally 0.373 ampere per square foot. This density was too high except when the rate of circulation of the solution also was high. About 50 mg. of fine silver dust came off in washing, and only 10 mg. were found in the clean-up box. The final solution contained 0.41 per cent KCy and 3.55 mg. of silver per 100 c. c. The cathodes were apparently stripped clean. The amount of silver recovered from the cathodes was as follows:

Quantity of silver recovered from cathodes.

	Grams.
From 1st set (4×5) of cathodes.....	7.700
From 2nd set (4×5) of cathodes.....	5.977
From 3rd set (2×10) of cathodes.....	2.258
Total recovery	15.935

The solution treated in the deposition box contained 15.040 grams of silver, giving an excess recovered of 0.895 gram, which came from the original cathodes that were not entirely cleaned in the previous test. The silver recovered was entirely free from gold, copper, or iron, and was decidedly fine. The silver dissolved in nitric acid, leaving no residue, and the nitric acid solution gave no color test with ammonia.

THE USE OF THE CLEAN-UP BOX WITH RAPID CIRCULATION.

The capacity of the deposition box had now been so increased that it far exceeded that of the clean-up box, and I next tried to increase the capacity of the latter in the way that I had that of the deposition box. This was not easy, as it was desirable to use sheet cathodes from which the metal could be stripped.

The sheet-iron cathodes of the clean-up box were of two types, which will be designated as A and B. The surface of the type-A

cathodes exposed to the solution was 7.5 by 8.5 cm. These cathodes touched the bottom of the box, and were perforated near the top with four 6-mm. holes to allow the solution to pass through. The total submerged area of the six type-A cathodes was 765 square centimeters. The six type-B cathodes used were each 7.5 by 8 cm. in size. They were so placed that they did not quite touch the bottom of the box, in order to force the solution to pass under. The area of these six cathodes on both sides was 720 square centimeters. The total cathode area was thus 1,485 square centimeters, or a little less than one-sixth square meter, or about 1.5 square feet.

The vertical edges of the cathodes A and B were insulated by ebonite strips 1 cm. wide and 6 mm. thick, which were placed on each side of the plates to stiffen them and to prevent their touching the electrodes to be stripped. The plates were then rubbed with a mixture of graphite and vaseline, just enough vaseline being used to make the graphite adhere. This coating was rubbed to a dry finish and the cathodes A and B were placed in alternate positions to force a proper circulation of the solution. Between the 12 cathodes were placed the wire-cloth anodes to be stripped, which had been the cathodes of the deposition box. Thirteen to 26 of these could be stripped at a time. In the first experiment 14 were used. The solution was circulated rapidly through the box by a small centrifugal pump.

The 14 wire cathodes in bunches of five, to be stripped, had been used before and had an unknown quantity of silver deposited on them. In the present test four different lots of dilute cyanide solutions, containing silver, were deposited on them in the usual way. The depositions were continued at convenient intervals for several days, the cathodes being taken out and dried between times. The conditions of deposition were unfavorable to good precipitation, on account of the local action between the iron and the silver during the drying, which would not occur in continuous practice. In all 55.328 grams of silver was calculated to have been deposited on the cathodes in these four runs, the exact amount, however, being greater on account of the unknown amount on them in the beginning.

Stripping followed with one small "Nungesser" cell (zinc and copper oxide in NaHO) which gave a voltage of 0.75 in open circuit, but much less on a closed circuit. At first, with 12 cathodes and 14 anodes to be stripped, the cell gave 0.4 volt and 1.4 amperes on a closed circuit. This soon fell to 0.3 volt, but the current remained at slightly more than 1 ampere till the electrodes were nearly stripped (five hours), when the voltage rose to 0.58 and the amperage fell to 0.38. In seven and one-half hours the wire electrodes were apparently completely stripped, the voltage of the cell having risen

to 0.68 volt and the current having fallen to 0.2 ampere, the current density being 8.0 amperes per square meter at the start and about 1.11 at the end of the stripping. The silver formed a brilliant white deposit and adhered firmly. The only trouble was a tendency for the silver to creep on the edges of the ebonite strips where a little graphite had accidentally adhered.

The stripped anodes were then removed, washed, and dried, and the process of electroconcentration was again undertaken. The six type-A electrodes were now made anodes, and the B electrodes were made the cathodes. The voltage was varied from 0.2 volt at the start to 0.68 at the end. The current at the start was 0.96 ampere and at the end 0.062. The cell was run 22 hours, when the silver was found to be entirely removed and concentrated on the six B cathodes. It was found to be in beautiful coherent sheets that could be readily peeled off. The formation of these strips is a complete refutation of the statements of Prof. Neumann that this is impossible. The total weight of silver concentrated on the six B cathodes weighed 91.8 grams. The cyanide solution at first contained about 1 per cent KCy. To this more was added from time to time, and at the end of the test the content of free KCy was 0.41 per cent. The silver content at the end was 4.42 grams of silver in 1.25 liters of solution, or about 0.354 per cent. Of course it would have been easily possible to reduce this amount of residual silver to almost any smaller amount by electrolysis. In one test the silver content of the solution in the stripping box was reduced to 0.98 mg. per 100 c. c., the KCy content being 0.59 per cent. If the solution were to be discarded the silver value could be reduced to a few cents per ton by circulating the solution a few times through the deposition box.

These excellent results were repeatedly duplicated, and the good coating with this dense current would have been impossible without the rapid circulation of the stripping solution.

DEPOSITION OF GOLD ON IRON WIRE-CLOTH CATHODES.

The electrodeposition of gold from cyanide solution is both theoretically and practically more difficult than that of silver. Theoretically the difference is to be judged by the electromotive force determinations of these metals in cyanide solutions.* Thus in 6.5 per cent KCy solution gold is electropositive to silver by 0.100 volt; in 0.65 per cent KCy solution, by 0.07 volt; in 0.065 per cent KCy solution, by 0.035 volt; and in 0.0065 per cent KCy, by 0.030 volt. This latter strength of solution is near the critical point at which both gold and silver become electronegative to cyanide solutions. Beyond

* See Christy, S. B., electromotive force of metals in cyanide solutions: *Trans. Am. Inst. Min. Eng.*, vol. 30, 1900, pp. 921-2.

that point, while both metals remain electronegative, the silver becomes less so (less "noble") than does gold.

Practically the extent of the difference by which gold is less easily precipitated from cyanide solutions than is silver increases more than the figures would indicate. This is shown by a study of the following experiment.

DEPOSITION OF GOLD FROM GOLD CYANIDE SOLUTION.

At the beginning of the test there was 22 liters of gold cyanide solution, containing 0.198 per cent KCy, 0.1 per cent KHO, and 3.696 grams Au as KAuCy_2 . Fourteen bunches of five iron-wire cathodes were used. The flow was at the rate of 8 liters per minute, or 480 liters per hour. The mean area was 4 square feet. The results are shown below.

Results of electrodeposition experiment with gold cyanide solution.

Hours.	Volts.	Amperes.	KCy, per cent.	Gold, mg. per 100 c. c.
0	0	0	0.198	16.80
1	1.5	2.00 to 0.6	.190	14.59
2	1.5	0.61 to 0.52	.191 (?)	5.92
3 ^a	1.5	0.52 to .30	.188	1.40
4	1.5	1.02 to 0.43	.182	.09
5	1.5	0.41	.174	.02
6	1.5	0.41	.165	Trace.

^aNote interruption of current for half an hour. Here the electric current failed, cathodes were removed and drained. After delay of one-half hour the current came on and the cathodes were replaced. The current was high at first, then dropped rapidly.

The ampere-hour efficiency, assuming gold as univalent, 7.35 grams per ampere-hours, was as follows:

Data showing ampere-hour efficiency.

Hour.	Gold, mg. per liter.		Liters treated.	Gold precipitated, grams.	Average ampere-hours.	Theoretical gold, grams.	Ampere-hour efficiency.
	In sample.	Difference.					
0	168.0						<i>Per cent.</i>
1	145.9	22.1	21.79	0.4816	1.12	8.2320	5.79
2	59.2	86.7	21.68	1.8800	.57	4.1897	44.87
3	14.0	45.2	21.57	.9750	.46	3.3812	28.71
4	.9	13.1	21.46	.2811	.56	4.1162	6.83
5	.2	.7	21.35	.0149	.41	3.0137	.49
6	Trace.	.2	21.24	.0042	.41	3.0137	.0139
				3.6368		25.9465	14.01

In the above experiment, the initial volume, 22 liters, was reduced by 0.210 liter for the first sample and by 0.110 liter for each succeeding sample, taken at the end of each hour. Hence a deduction from the volume for the solution treated each hour is necessary. A num-

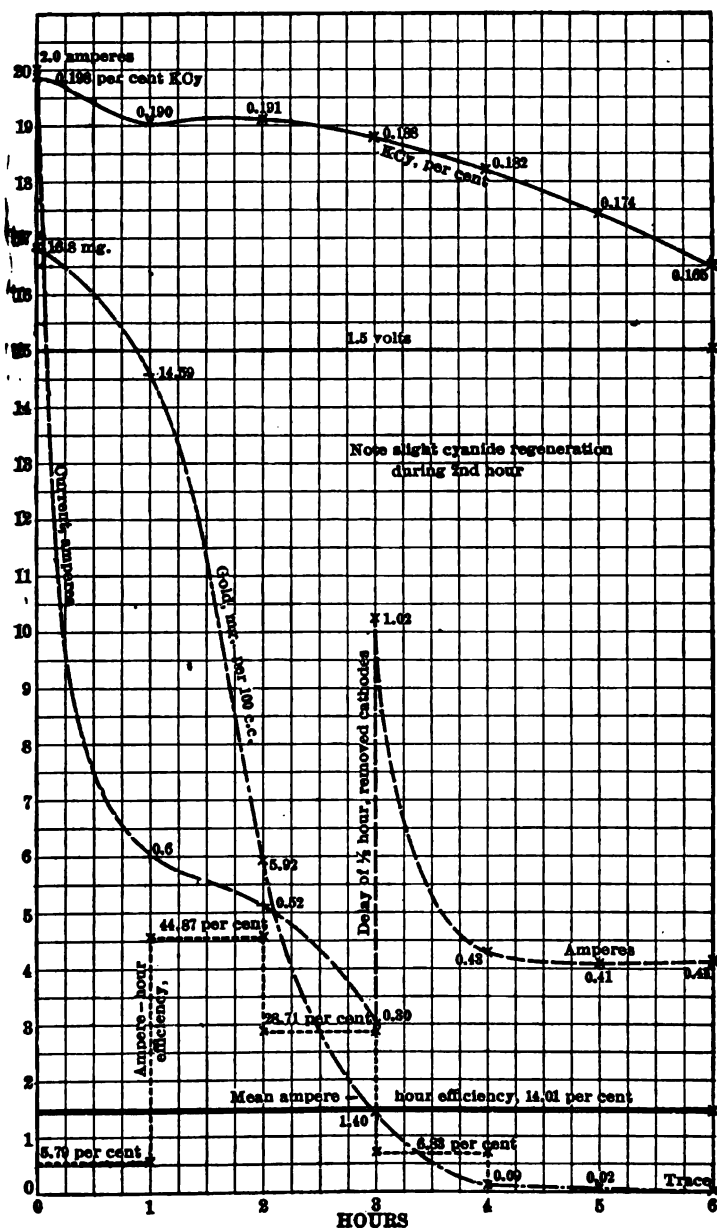


FIGURE 21.—Electrodeposition results with gold cyanide solution. Twenty-two liters of solution, containing 0.198 per cent KCy, 0.1 per cent KHO, and 3.696 grams of Au as KAuCy_2 , taken for test. Seventy 16-mesh wire-cloth cathodes; estimated area, both sides, 2 to 10 square feet. Rate of flow, 8 liters per minute; velocity through open box, 46 inches per minute; through cathodes, 105 inches. Note slight cyanide regeneration during second hour.

ber of important conclusions may be drawn from this experiment (see fig. 21).

First, note the low efficiency during the first hour. The cause of this low efficiency is that between the experiments, which were dis-

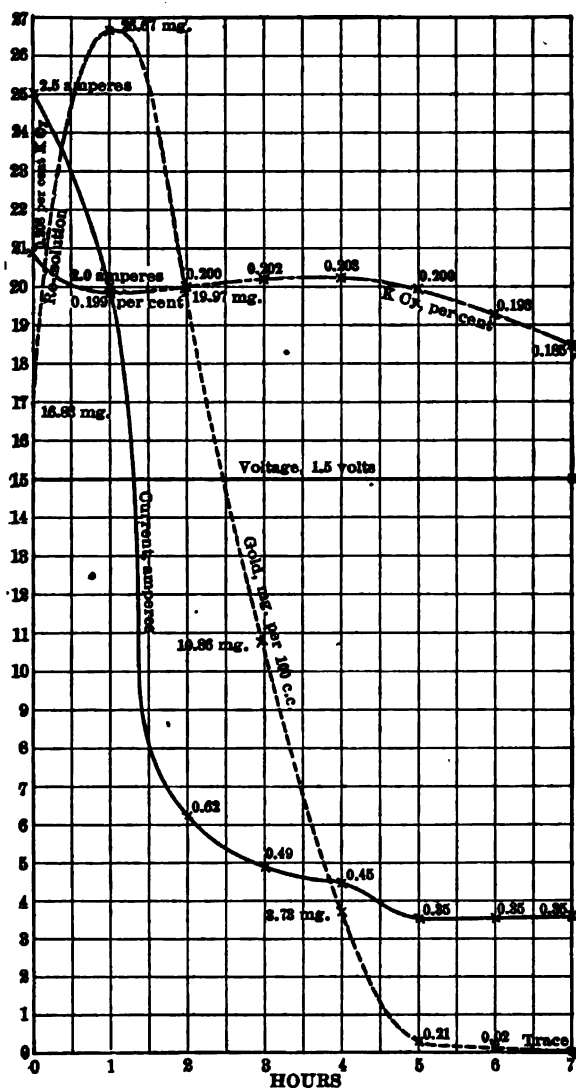


FIGURE 22.—Gold deposition results showing effect of local action on iron-wire cathodes. Eight carbon comb anodes; 70 16-mesh cathodes; estimated surface, between 2 and 10 square feet. Twenty-two liters of solution, containing 0.208 per cent KCy , and 16.83 mg. Au per 100 c. c. Rate of flow, 8 liters per minute; velocity, 46 inches per minute through open box and 105 through cathodes.

continuous, the iron-wire cathodes had rusted, and a large part of the current in the first hour was consumed in reducing the oxide. The

serious nature of this trouble is shown more clearly in later experiments.

Second, the high ampere-hour efficiency in the second hour (44.87 per cent) would have been still higher if the cathodes had been entirely clean the first hour. The ampere-hour efficiency for the entire six hours, 14.01 per cent, was remarkably high.

The extraction was good. In five hours the gold value of the solution was reduced to 12 cents per ton and in six hours to less than 6 cents. The precipitation is, however, much less rapid than with silver. The rate would have been increased by a higher voltage but at the cost of the ampere-hour efficiency.

A slight regeneration of cyanide during the second hour, when the gold precipitation was greatest, was noted.

THE EFFECT OF LOCAL ACTION ON IRON-WIRE CATHODES.

Attention has already been called to the local action that takes place in intermittent silver precipitation. With gold it is more marked. The serious nature of this difficulty is illustrated by the following experiment: The gold-coated cathodes were drained and allowed to dry over night. They were not dried by heat, as the unequal expansion of gold and iron might cause the coat to scale. However, local action resulted which oxidized the iron with disastrous results. For the first hour, gold dissolved in spite of the current, showing that the coating of gold had been undercut by the rust, thus insulating it from the iron cathode, and allowing it to dissolve. The second hour still shows the bad effect of the rust, although the deposition was fair. The details and results of the experiment (see fig. 22) which illustrates this are as follows:

There were employed eight new carbon anodes. The cathodes were the 14 bunches of fine 16-mesh wire-gauze screens, which had been partly coated with gold from the previous run. The total volume of the solution at the start was 22 liters; it contained 0.208 per cent KCy, 168.3 milligrams gold per liter, and 0.1 per cent KHO. The flow was 8 liters per minute, or 480 liters per hour. The total cathode area was between 1.57 and 7.85 square feet. The area of the composite wire cathodes is hard to calculate. The minimum area would be that of the section of the cloth, 0.112 square feet (counting both faces) for each of the 14 groups, or 1.57 square feet. The maximum would be 5 times 1.57, or 7.85 square feet. In time the entire cloth becomes covered with metal, but not all at once. The true mean would probably be about 4 square feet, a little below the average of 4.71 square feet. The ampere-hour efficiency in this experiment is shown in the following table:

Data showing ampere-hour efficiency in experiment.

Hour.	Gold, mg. per liter.	Difference.	Number of liters treated.	Gold precipitation, grams.	Average ampere-hours.	Theoretical gold, grams.	Ampere-hour efficiency, per cent.
0	188.3						
1	266.7	-98.4	21.79	-2.1442	2.100	15.434	Negative.
2	190.7	+67.0	21.68	+1.4525	1.570	11.540	12.58
3	108.6	+91.1	21.57	+1.9650	.826	3.866	50.81
4	37.3	+71.3	21.46	+1.5300	.478	3.513	43.55
5	2.1	+35.2	21.35	+ .7515	.406	2.964	25.18
6	.2	+ 1.9	21.24	+ .0404	.350	2.572	1.57
7	Trace.	+ .2	21.13	+ .0042	.350	2.572	.16
				+3.5994		42.481	

During the first hour the ampere-hour efficiency was negative, as nearly two-thirds of the gold already precipitated was redissolved; but in the next hour the current had begun to cure the evil, and 12.58 per cent of the current was utilized. During the next hour the efficiency had risen to 50.81 per cent, slowly falling to a fraction of 1 per cent. Besides the dissolved gold the new gold, amounting to 3.6842 grams, was practically all precipitated. However, considering the net gold only, the efficiency is high. Dividing 3.5994 by 42.481 (the weight of gold that the current should have precipitated) gives for the whole period 8.47 per cent, which is a good result in spite of the poor start.

To be sure that the carbon anodes had not prevented the solution (containing 0.185 per cent KCy) from dissolving gold, 100 c. c. of the tailing solution was rotated for half an hour in a $\frac{1}{2}$ -liter flask filled with air to aerate it, and then was rotated with a gold strip in it for one hour. The quantity of gold thus dissolved was 2.47 mg. A parallel experiment with pure KCy (0.185 per cent) solution gave 2.41 mg. Hence the carbon anodes had not prevented solution of gold.

The same wire-cloth cathodes, 14 groups of five, from the above test with gold still on them were again treated with practically the same results. (See figure 23). Eight of the hard-carbon anodes were used. The cathode area was between 1.57 and 7.85 square feet. The rate of flow was 8 liters per minute. The ampere-hour efficiency under the above conditions is shown in the following table:

Data showing ampere-hour efficiency of experiment.

Hour.	Gold, mg. per liter.		Number of liters treated.	Gold precipitation, grams.	Average ampere-hours.	Theoretical gold, grams.	Ampere-hour efficiency, per cent.
	At start.	Difference.					
0	188.3						
1	270.4	-101.1 (?)	21.79	-2.2080 (?)	0.566	4.175	Negative.
2	213.9	+ 51.5	21.68	+1.1165	.410	3.017	37.01
3	165.5	+ 63.4	21.57	+1.1513	.410	3.017	38.18
4	94.0	+ 71.5	21.46	+1.5248	.430	3.161	48.24
5	29.3	+ 64.7	21.35	+1.3812	.434	3.190	43.30
6	2.0	+ 27.3	21.24	+ .5796	.382	2.807	20.66
7	.2	+ 1.8	21.13	+ .0380	.345	2.536	1.50

The net amount of gold precipitated was 3.5891 grams, whereas the current should have precipitated 21.903 grams. Therefore, the total ampere-hour efficiency was 16.39 per cent, and in spite of the poor

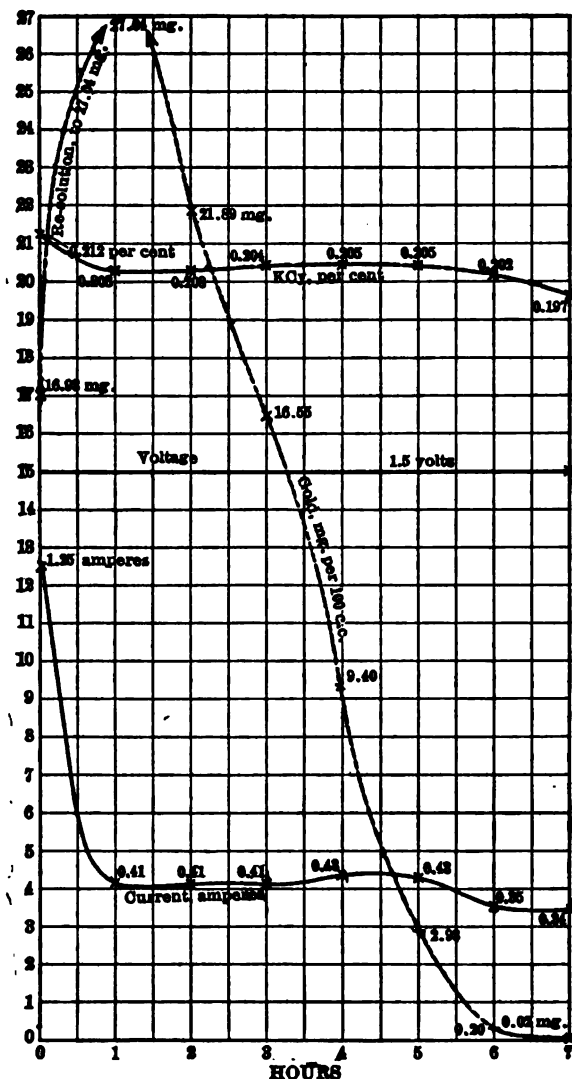


FIGURE 23.—Electrodeposition results showing re-solution of gold due to rusting of cathodes. Eight "Electra" carbon comb anodes. Seventy 16-mesh wire-gauze cathodes, in 14 groups of 5 each; estimated total surface area, between 2 and 10 square feet. Solution, 22 liters, contained 0.212 per cent KCy, 0.1 per cent KHO, and 16.98 mg. Au per 100 c. c. Rate of flow, 8 liters per minute; velocity through open box, 46 inches per minute; through cathodes, 105. Note fall of cyanide during solution of gold and slight rise during fall of gold.

start was decidedly satisfactory for the whole period. The current was rather low because the solution was a new one used for the first

time, and did not contain so many dissolved salts; the current was relatively better after a while because the contacts were better. The fall of the cyanide during the solution of the gold and the slight rise during its rapid precipitation should be noted. The rise was less than the fall, owing to continuous oxidation.

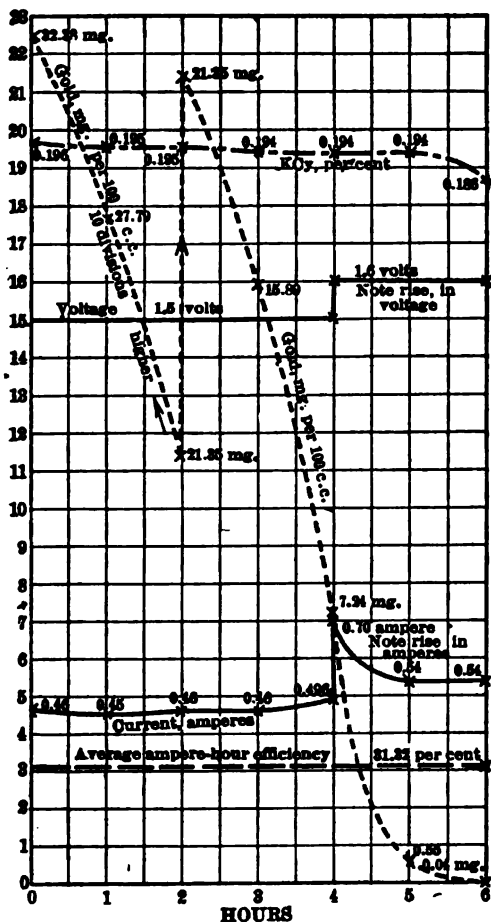


FIGURE 24.—Electrodeposition results with rich gold solution. Solution, 21.68 liters, containing 32.48 mg. Au per 100 c. c. Six carbon comb anodes; seventy (14 groups of 5) 16-mesh wire-cloth cathodes; total estimated surface area, between 2 and 10 square feet. Rate of flow, 8 liters per minute or 480 per hour; velocity through open box, 46 inches per minute; through cathodes, 105.

The next experiment was made with 14 bunches of five cathodes of 16-mesh wire cloth. The total area of these was estimated to be between 1.57 to 7.85 square feet. They were gold coated from previous use. Six carbon anodes were used. The solution contained 0.205 per cent KCy and 0.1 per cent KHO. The flow was 8 liters per minute. The solution at the start contained 16.92 mg. gold per liter,

but owing to bad contacts, the gold dissolved, so that the content in the solution increased to 32.38 mg. The ampere-hour efficiency under these conditions (see also fig. 24) is shown in the following table:

Data showing ampere-hour efficiency in experiment.

Hour.	Gold, mg. per liter.		Number of liters treated.	Gold precipitated, grams.	Average ampere-hours.	Theoretical gold, grams.	Ampere-hour efficiency, per cent.
	At start.	Difference.					
0	323.8						
1	277.9	45.9	21.68	0.9951	0.45	2.307	30.08
2	213.5	64.4	21.57	1.3390	.46	3.332	41.07
3	153.9	59.6	21.46	1.1717	.46	3.332	34.65
4	72.4	81.5	21.35	1.8487	.496	3.645	50.68
5	5.5	66.9	21.24	1.4209	.805	4.447	31.95
6	0.4	5.1	21.13	0.1078	.540	3.999	2.72

The average efficiency for the whole period is thus shown to be 31.32 per cent, a result which certainly is very satisfactory.

DEPOSITION OF GOLD FROM DILUTE SOLUTION ON RUSTED IRON CATHODES.

The solution consisted of approximately 22 liters of 0.047 per cent cyanide, containing 1.010 mg. of gold per 100 c. c. Seventy (14 sets of five) coarse wire cathodes were used, the area being between 1.57 and 7.85 square feet. The rate of flow was 7 liters per minute, or 420 liters per hour. The results of this experiment are graphically shown in figure 25.

The cathodes had been used before and had rusted, with an unknown amount of gold on them. During the first hour, owing to the extent of the rusting, the gold stripped off the cathodes and the content in the solution increased to 2.46 mg. per 100 c. c. As stripping would not occur in continuous practice, the analysis begins with this content of gold in the solution. At the close of the experiment all the cathodes except two were coated with gold, but the coating was irregular, showing the effect of the rust. Near the surface, where air had access, the coating was thinner than below. The carbon anode at 1.6 volts did not show any signs of corrosion. The ampere-hour efficiency attained in this experiment is as follows:

Data showing ampere-hour efficiency attained.

Hour.	Gold, grams per liter.		Number of liters treated.	Gold precipitated, grams.	Average ampere-hours.	Theoretical gold, grams.	Ampere-hour efficiency, per cent.
	At Start.	Difference.					
0	0.0246						
1	.0234	0.0012	21.68	0.0290	0.52	3.324	0.65
2	.0210	.0024	21.57	.0518	.46	3.281	1.53
3	.0075	.0135	21.46	.2800	.36	2.646	10.96
4	.0061	.0024	21.35	.0512	.34	2.499	2.05
5	.0023	.0023	21.24	.0695	.33	2.353	2.58

This table shows the detrimental effect of rust on iron-wire cathodes. The gold that had been previously deposited on the cathodes dissolved in spite of the current. It is possible that oxide of iron had formed under the gold and insulated it from the iron, thus enabling the gold to redissolve. The detrimental effect of the rust continued during the next hour and the ampere-hour efficiency was only 0.68 per cent. In the next hour it had risen to 1.53 per cent, and in the next hour to 10.96 per cent. The current, then, is capable of curing the evil, and for this purpose I have devised what I call the "hospital cell," mentioned on page 113.

A test which further shows the effect of rust upon cathodes is the following: Seventy (14 sets of five) gold-coated rusted iron-wire cathodes were used in a solution containing about 1 mg. gold per 100 c. c., or \$6.03 a ton. The cathode surface was between 1.57 and 7.85 square feet. The flow was 7 liters per minute. The experiment was begun with six perforated platinum anodes. The pressure was 2.5 volts. The current at first was 1.33 amperes, falling in four hours to 0.91 ampere. The gold on the rusted iron wires dissolved in spite of the current. The solution was then made up to 22 liters, when it contained 4.12 mg. gold per 100 c. c. and 0.42 per cent KCy. When six perforated and peroxidized-lead anodes were used the voltage re-

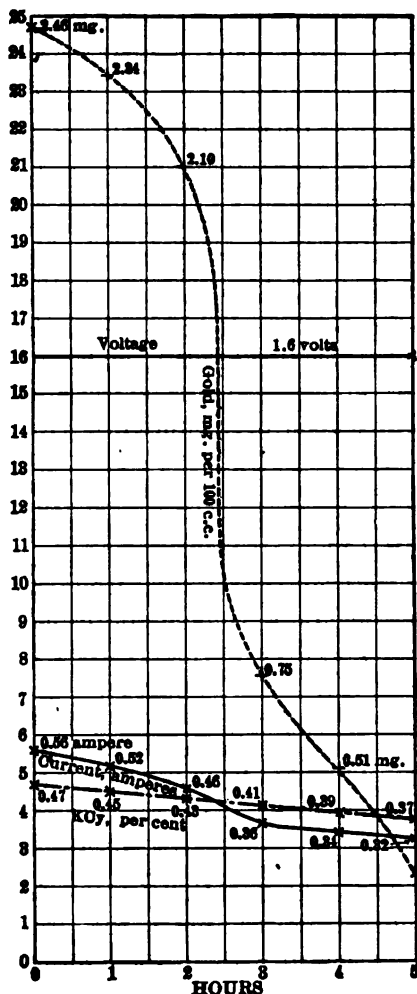


FIGURE 25.—Electrodeposition results with dilute gold solution on rusted iron-wire cathodes. Solution, 21.68 liters, contained 0.047 per cent KCy, 0.1 per cent KHO, and 2.46 mg. Au per 100 c. c. Seventy (14 sets of five) 16-mesh wire-cloth cathodes; estimated surface area, between 2 and 10 square feet. Rate of flow, 7 liters per minute, or 420 per hour; velocity through open box, 41 inches per minute; through cathodes, 91. No rise in cyanide was shown.

mained unchanged, but immediately the current rose to 3.65 amperes. The results are graphically shown in figure 26. The current efficiency was as follows:

Data showing ampere-hour efficiency in test.

Hour.	Gold, grams per liter.		Number of liters treated.	Gold precipitated, grams.	Average ampere-hours.	Theoretical gold, grams.	Ampere-hour efficiency, per cent.
	At start.	Difference.					
0	41.2						
1	2.5	38.7	22.00	0.85140	3.07	22.563	3.77
2	0.5	2.0	21.89	0.04378	3.59	26.240	0.17
3	Trace.	0.5	21.78	0.01099	3.55	26.098	0.04

The precipitation is at the rate of about 176 tons per day in a 1-ton box, much better than that obtained by Siemens & Halske. Moreover, this result is evidently all that could be desired, but it was accomplished at the expense of the ampere-hour efficiency. Of course good precipitation is vital, and is more important than electrical efficiency. This would have been higher, except for the rusted condition of the cathodes. The great importance of overcoming the difficulty from rusty cathodes is evident, and will be specially considered later.

Most of these experiments with gold solutions were conducted at too low a voltage for rapid and complete precipitation. Low voltages were purposely used to find the lowest limit that would give high electrical efficiency and satisfactory regeneration of cyanide. With the ordinary gold solutions used in mill practice the cyanide regeneration is so small that it is obviously better to run at a higher voltage, at least 2.5 volts, to insure rapid and complete precipitation.

EXAMPLE OF A GOLD CLEAN-UP.

The next problem for consideration is that of recovering the gold deposited upon the iron wire-cloth cathodes. Fourteen "bunches of five" of these cathodes had been coated with about 14.7 grams of gold in the experiments just described.

Six flat, sheet-iron, graphite-coated cathodes were inserted in the box along with the 14 cathodes to be stripped and now serving as anodes. A 0.97 per cent potassium cyanide solution was circulated in the stripping box by means of the centrifugal pump. A "Nungesser" cell was used to supply the electrolytic current. The voltage, which at first was 0.75 volt, decreased in one hour to 0.39, then, as the stripping progressed, gradually rose to 0.69 in six hours. The gold was thus concentrated on both sides of six sheet-iron cathodes. As the film was very thin, however, three of the cathodes were made anodes and the gold was all concentrated on three cathodes.

The Nungesser cell gave at first 0.50 volt and 0.26 ampere. After

being allowed to run 16 hours over night, the cell gave in the morning a voltage of 0.70, and the current had fallen to 0.001 ampere. The three cathodes were coated on both sides with a beautiful clear yellow coat of metallic gold which could be easily stripped from the graphite-coated iron in a tough coherent film, the gold thus recovered weighing 6.069 grams. The cyanide content was reduced to 0.57 per cent during this deposition.

The solution from the clean-up box contained 329.05 mg. gold per 100 c. c. (one-third per cent), thus the 1,800 c. c. held 5.923 grams. Loosely adhering to the cathodes was some fine gold which weighed 130 mg. Hence there was recovered in sheet metal, by a voltage of less than one volt, 6.069 grams of sheet gold and 0.130 gram of scrap. The amount of gold in solution not precipitated was 5.923 grams, making a total weight of 12.122 grams. Unfortunately in starting the experiment part of the rich solution was lost so that a check was not possible.

Evidently, to recover all the gold a higher voltage, say 1.6 or 2.5 volts, would have been necessary. Complete precipitation and recovery of the gold by electrolysis could be made at any time. This has been done time and time again,

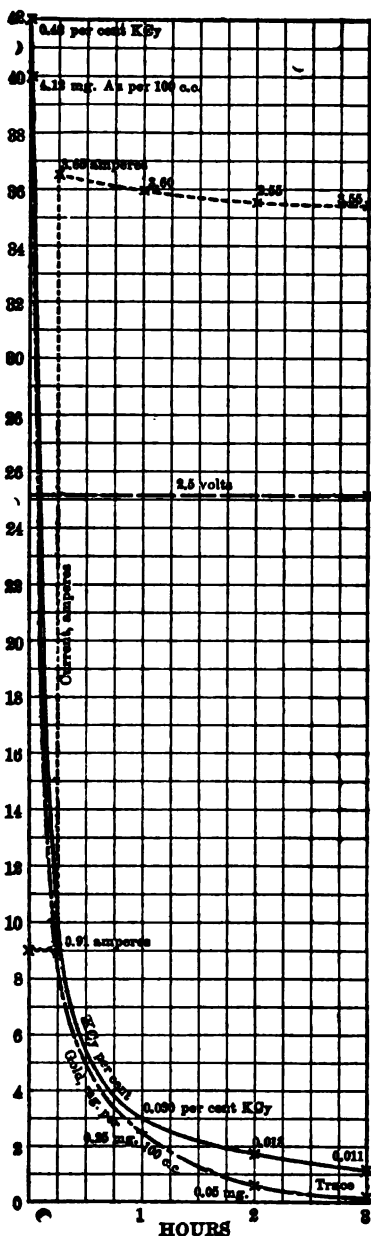


FIGURE 26.—Electrodeposition results showing effect of rust on cathodes. Seventy (14 sets of five) 16-mesh gold-coated rusted wire-cloth cathodes used; estimated surface area, between 2 and 10 square feet. Solution, 22 liters, contained 0.42 per cent KO₂, 0.1 per cent KNO₃, and 4.12 mg. Au per 100 c. c. Rate of flow, 7 liters per minute or 420 liters per hour; velocity through open box, 41 inches per minute; through cathodes, 91. For first quarter hour, 6 platinum anodes used; current was only 0.91 ampere; then 6 perforated peroxidised (PbO₂) lead anodes were used; current rose to 3.65 amperes. Note the better precipitation at higher current, but greater and continuous destruction of cyanide.

so there is no doubt on that point. The sheets of gold adhering to the cathodes obtained here are a complete refutation of the claim of Prof. Neumann that recovery of the gold in this manner is impossible.

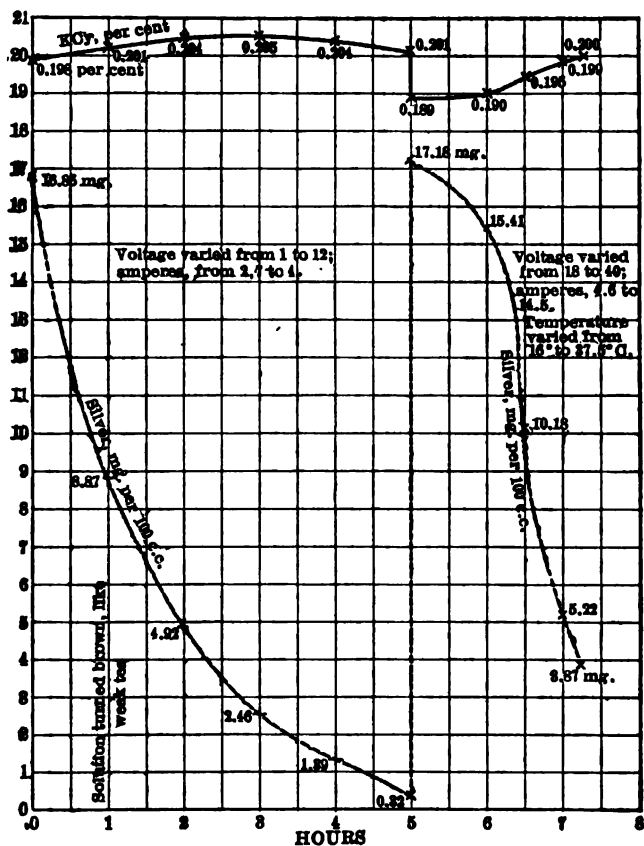


FIGURE 27.—Results of silver electrodeposition with cathodes and anodes horizontal. One perforated sheet-iron anode. One set of five 30-mesh iron wire-cloth cathodes, each 10 by 8.5 cm.; total area of pervious part, 81 square cm.; area when covered with silver was indeterminate. Rate of flow, 11 liters per minute or 660 per hour; velocity through open box, 53 inches per minute; through cathodes, 120.

RECOVERY OF SILVER ON HORIZONTAL PERVIOUS CATHODES.

All the electrodes previously considered were placed vertically in the deposition box. It was now thought wise to see what could be done in silver recovery by using horizontal pervious cathodes, with removable pervious anodes above them. If this arrangement were successful, a clean-up would be unnecessary, for the silver could be scooped up with a shovel. Such a construction is not so favorable for deposition as the previous arrangement, for only one anode

cathode. The cathode was 10 by 8.5 cm. in size, only 81 sq. cm. of which was pervious. The voltage, which varied, began at about 3 and finally was raised to 40 volts. The rate of flow was 11 liters per minute, or 660 liters per hour. The results are shown in figure 27. At the end of the second hour the solution began to have a slight brownish cast, which gradually increased, till at last it was dark, reddish brown, about like black-tea infusion. Its temperature had risen to 37.5° C. The room temperature was 20° C. At the end of the fifth hour 3.76 grams of Ag as KAgCy_2 was added and 1.3 liters of water to make the total volume up to 22 liters. The silver was deposited as a loose, gray sponge on the wire gauze, which was coated only on the top.

Even if boxes were separately arranged in cascade one over another, it is probable that sufficient cathode area to render the method practicable could not be obtained. Such high voltages as were used are, of course, out of the question, but the clean-up would be simple (the sponge was a beautiful, feltlike network, pervious to the solution). All that would be necessary in practice would be to remove the silver with a shovel.

EXPERIMENT WITH A HORIZONTAL ELECTROLYTIC FILTER.

An experiment was made with a horizontal electrolytic filter. In this apparatus the anode, of perforated platinum, was at the top. After three hours, 12 turns of No. 18 platinum wire were used. The cathodes were of wire cloth such as had been used previously, and were each 10 by 8.5 cm. in dimension. The solution, 22 liters, contained 0.211 per cent KC_y and 0.1 per cent KHO . The flow was at the rate of 10 to 12 liters per minute. The results are shown in figure 28.

At the end of the second hour the experiment was temporarily stopped, and the cathodes were removed during the noon hour at that time. At the end of the third hour the perforated platinum sheet was removed and replaced with a platinum wire anode. This wire anode allowed a better escape of the gas from the anode and gave a stronger current at a lower voltage. The ampere-hour efficiency is shown in the following table:

Data showing ampere-hour efficiency in experiment with horizontal filter.

Hour.	Silver content, mg. per liter.		Number of liters treated.	Silver precipitated, grams.	Ampere-hours.	Theoretical silver, grams.	Ampere-hour efficiency.
	At start.	Difference.					
0	1138.4						
1	683.5	454.9	21.79	9.816	11.0	44.370	22.17
2	263.5	420.0	21.68	9.882	11.2	45.080	20.70
3	96.2	167.3	21.57	8.909	11.3	45.480	7.94
4	25.0	61.2	21.46	1.813	13.1	52.724	2.47
5	6.8	18.2	21.35	0.382	13.0	52.394	0.73

The ampere-hour efficiency was high notwithstanding the wastefully high voltage used; it averaged 10.19 per cent for the entire period. Such high efficiency would have been impossible except for the rapid circulation. The metal came down as a beautiful white, porous layer of spongy silver, $\frac{1}{4}$ inch thick, of such quality as to permit it to be scooped up with a shovel. This might be a convenient modification of the clean-up box, but some material for the "cathode-anodes" would have to be used that would not be corroded after the current were stopped—excelsior charcoal, for instance.

REGENERATION OF CYANIDE.

During the first three hours there was a remarkable regeneration of cyanide, as is shown in the following table:

Data showing regeneration of cyanide.

Hour.	KCy content.	Increase.	Volume treated.	KCy regenerated.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Liters.</i>	<i>Grams.</i>
0.....	0.210			
1.....	.265	+0.055	21.79	+11.593
2.....	.313	+ .048	21.68	+10.406
3.....	.332	+ .019	21.57	+ 4.068
4.....	.329	a- .003	21.46	a- .644
5.....	.320	a- .009	21.36	a- 1.922

a Loss.

The total number of grams regenerated in three hours is seen to have been 26.097. The ratio of cyanide regenerated to the silver precipitated was as follows:

$$\text{First hour.} \quad \frac{\text{Weight of KCy}}{\text{Weight of Ag.}} = \frac{11.593}{9.816} = 1.181.$$

$$\text{Second hour.} \quad \frac{10.406}{9.332} = 1.121.$$

$$\text{Third hour.} \quad \frac{4.098}{3.609} = 1.135.$$

Suppose that for each gram-atom of silver precipitated (molecular weight, 108), two gram-molecules of KCy (molecular weight, 130) were regenerated; then for each gram of silver 1.2037 grams of KCy would be regenerated at the cathode. In this experiment conditions were more favorable for reduction at the cathode than for oxidation at the anode, and the highest ratio attained was very nearly, but not quite, the maximum that would have been possible had there been no oxidation at the anode. It would seem, then, either that the great current density at the anode tended to prevent oxidation, or that the lesser density at the cathode favored reduction of the cyanate formed at the anode. Probably both tendencies acted together to produce this favorable result.

ANODES.

"Oh, for an anode!" There is no one who has done much electrolytic work who has not uttered this exclamation from the bottom of his soul.

SOLUBLE ANODES.

The idea of using soluble anodes so as to avail one's self of the solution energy of the soluble anode is at first thought a perfectly natural one, but on working over the field, only two soluble metals are available, zinc and aluminum. Logically, iron may be considered as a soluble anode; but for convenience, anodes of iron are discussed under the heading "Insoluble Anodes" (pp. 98 to 108).

ZINC AND ALUMINUM.

If commercial zinc is used as an anode, the gold and the silver are precipitated on the anode by local action, even more than on the cathode. Hence both anode and cathode have to be cleaned. To prevent local action on the anode, the use of amalgamated zinc plates for anodes has been proposed by Croasdale.* But unless the anodes are made very thick and heavy and expensive, they become brittle when amalgamated and fall to pieces. The large area necessary in treating dilute gold solutions is hard to obtain. The same objections apply with even greater force to the use of soluble aluminum anodes. Difficulties of ore leaching, caused from the gelatinous hydrate of aluminum, are further complications, and quicksilver rots aluminum plates even more fatally than zinc.

I think it is safe to take the position that if zinc or aluminum are to be used to precipitate gold or silver from cyanide solutions, they should be used alone and without external electricity, as the combination of these metals with applied electric energy does not seem to be a happy one.

INSOLUBLE ANODES.

Insoluble anodes, of course, add no energy to the bath, and offer an increased resistance to the current owing to polarization effects. Nevertheless in treating ore solutions such anodes would be better than soluble ones if they really could be obtained absolutely insoluble and permanent in daily use.

PLATINUM.

Platinum, if it were cheap, would solve most of the problems; but except for experimental work and for certain rare uses its high cost puts it entirely out of the question.

* Croasdale, Stuart, Electrolytic precipitation of gold from cyanide solutions: Eng. and Min. Jour., Dec. 12, 1896, vol. 62, pp. 557-558.

HARD CARBON.

Hard electric-light carbon serves admirably in cyanide solutions when the voltage is kept below that at which water is decomposed. But this point is so low (1.48 volts) that it is below the voltage necessary to use in practice. At higher voltages the carbon is slowly acted upon, and the solution becomes colored like weak tea. I have found, however, by repeated experiments that even when darkly colored such cyanide solution, when aerated, will dissolve gold as freely as fresh cyanide solution of equal strength. This form of carbon resists the chlorides very well but soon succumbs to alkaline sulphates, which are sure to be present in cyanide solutions after treating sulphide ores and must be reckoned with. I have tried to increase the resistance of this form of carbon by boiling for several days in vaseline or paraffin the anodes made of it, in order to confine corrosive action to the surface, but these efforts have been without success.

ACHESON GRAPHITE.

Graphite anodes, as produced by Acheson, of Niagara Falls, in the electric furnace, are a great improvement on the ordinary hard carbon. The substance is a better conductor than ordinary carbon and is much more resistant, but in practice it, too, fails when used with solutions containing soluble sulphates. As already mentioned in the experience with Acheson graphite anodes in the Oliver process, at the North Star mine, as long as the process was confined to the treatment of solutions coming from sands that contained very little sulphate, the anodes stood up very well, but as soon as solutions coming from the concentrated sulphides were treated, the abundant sulphates soon reached the weak points of the anodes and they disintegrated finally into a soft black mud. This failure was one of the causes that led to the abandonment of the entire process. The ease with which the Acheson graphite can be machined in a lathe and built up into many forms is a great advantage. I have used it in my experiments in the form of "comb anodes." These are made by screwing four $\frac{3}{8}$ -inch graphite rods, so as to project $4\frac{1}{2}$ inches, into the rectangular block or base bar, also of graphite. Such anodes are rather brittle and have to be handled carefully, but if sulphate solutions are avoided they will last for years. There is never any doubt as to the security of the connections. I usually solder a copper wire to a brass ring, which tightly fits around one of the projecting rods; then the anode requires no further attention, except to avoid dropping it. Enough has been said to show that the Acheson graphite anode can not be said to solve the entire problem.

PEROXIDIZED-LEAD ANODES.

The discovery by Andreoli that peroxidized-lead anodes could be used in cyanide solutions has been of great value to the art. It does not seem to be a matter of indifference how these anodes are prepared. It would seem as if those prepared either as is done by Mr. Butters—namely, by scratching the surface with a stiff brush and then making them anodes in a 1 per cent solution of permanganate of potassium for at least one hour, employing a current density of 1 ampere per square foot of anode—or else by making them first for a short time cathodes and then anodes in a caustic potash solution containing peroxide of lead, give better results with cyanides than those peroxidized in sulphuric acid. However, it is not absolutely certain which is the better way, as enough work has not been done on the subject.

Well prepared, these anodes last about a year. Sooner or later they begin to show white, downy, fungus-like growths that form usually in vertical streaks, which extend up and down the plate. If these growths are cleaned away, naked lead is seen to have been exposed to corrosion. "Local action" has started and deep depressions are formed in the plate, which soon disintegrates. Prof. Kerns says that these white formations are composed of the hydrate of lead. That they often contain lead cyanide I have proved by the following experiment.

Some of the white formation was washed by decantation till the washings failed to give any reaction for cyanide of potassium. The residue was then placed in a flask, acidified with sulphuric acid, and air was blown through the mixture and allowed to bubble through a 1 per cent solution of caustic potash. In a short time a distinct reaction was obtainable for cyanide in the alkaline solution.

It would appear that the substance is a varying mixture of hydrate, carbonate, and cyanide of lead. In time such a growth sometimes becomes slightly peroxidized on the surface, but once it has started, eats like a cancer through the plate. Prof. Kearns suggests that the cause of these growths starting is the porosity of the peroxide film. This is most likely; but I have seen growths start on thick coatings where the occurrence of pores was unlikely. Seemingly a frequent cause may be the cracking of the rather brittle oxide when the plastic lead beneath it is bent. The films on these long lead strips may be easily cracked by bending of the sheets as they hang loose in the bath, and particularly by their being handled.

Another possibility is that under certain, but as yet unknown conditions of current density (accidental short circuits, perhaps) and temperature, the anion AuCy_2 or its components AuCy and Cy may act as reducing agents on the PbO_2 , thus exposing naked lead to the

action of the current. A similar reaction was encountered when I once tried to dry out a mixture of PbO_2 and glycerin on the steam bath. The mixture was decomposed with almost explosive violence and abundant minute shots of metallic lead were reduced from the mixture.

The flat shape of the ordinary sheet anode renders it peculiarly sensitive to a rupture of the protecting surface by bending. In order to meet this difficulty I have devised an improved anode (U. S. patent No. 883170, Mar. 31, 1908).

For this purpose I construct a frame of wood, shown in vertical longitudinal section at *A*, and in vertical cross section at *B*, in figure 29. If the wooden frame, *a*, is made large it should be strengthened by horizontal cross bars *b*. Strips of lead wire, *c*, one-eighth to one-fourth of an inch thick, and of cylindrical form are drawn through suitable holes bored in the horizontal bars of the frame, spaced one-half to one inch or more, thus stretching the lead wires, like the wires of a harp, alternately up and down through the frame. The wires are connected on the top by a lead bus bar, *d*, large enough to carry the total electric current, the individual wires being so proportioned as to carry their part of the electric current, which they must do without overheating. I prefer to protect the wood from the solution by painting it with the so-called "P & B" paraffin paint or some other similar substance. The bottoms of the lead wires are kept from touching the bottom of the box by placing them in a groove in the lower horizontal cross bar. Where the wires pass over the lead bus bar at the top they are secured to the bus bar, preferably by the process of lead burning rather than soldering, although soldering may be used if necessary. The end of the bus bar is connected at *e* with a copper wire, which connects with the main bus bar or the conductor introducing the positive current. After the anodes are thus constructed, the lead wires are prepared as follows:

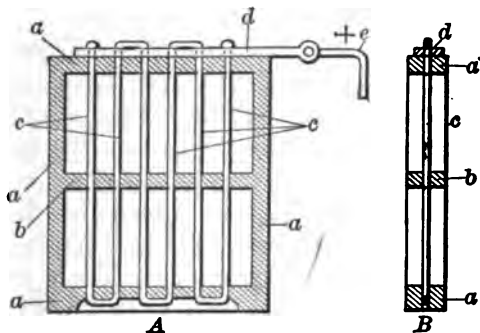


FIGURE 29.—Anode frame designed to prevent bending of metal. *A*, vertical longitudinal section; *B*, vertical cross section. *a*, Wooden frame; *b*, cross bars; *c*, lead wire; *d*, lead bus bar; *e*, copper wire connection to bus bar.

In order to get a clean metallic surface on the outside of the wires free from all insulating substances, I connect the wire with

the negative pole of a suitable source of electrical energy, making the lead wires cathodes for a short time in a solution of plumbate of soda. Any oxide or carbonate or hydrate that may have been formed on the outside of the lead wires becomes thus reduced to the metallic state, and an even surface is produced on the outside of the wire of lead. Without waiting for the film to oxidize in any way, the electric current is immediately reversed and the lead wires are made anodes in the same solution and are kept there until a firm, adherent, dark chocolate brown coating of peroxide of lead forms over every portion of the wire exposed to the action of the current. When this coating is of sufficient thickness to protect the wire, it forms a cylindrical film or tube of firmly adherent peroxide of lead. The anodes are then ready to be inserted into the bath of copper sulphate, cyanide of gold or of silver, or other solution to be treated, to act as pervious peroxidized-lead anodes. These peroxide-coated lead wires may also be replaced by solid rods of lead peroxide similarly arranged in a suitable frame.

Solid rods of peroxide of lead are not so good conductors as those with a lead core, hence they have to be made larger in section to carry the same current. They are, however, more durable. The wooden frame serves to protect the anode rods both from deformation and from destruction by short circuits. The tube-like coating of PbO_2 upon the lead wires is less likely to scale than the coating on flat moving sheets. With sheets the slightest deformation is likely to cause the coating to scale. The wire anodes thus described expose less surface to the bath than the flat anodes. They thus give a lower oxidizing effect, an advantage which is highly desirable in treating cyanide solutions.

It is hard to obtain data on the conductivity of lead peroxide. Landolt and Börnstein* give the following for "hydrated lead peroxide":

Conductivity of "hydrated-lead peroxide."

	Conductivity, per c. c.	Resistance, ohms per c. c.
	0.163	6.13 (Shields)
	4.68	0.214 (Weyde)
With 1.5 per cent H_2O -----	335.00	0.00298 (Ferchland)
Pressed powder-----	0.435	2.3 (Sheintz)

The conductivity of lead, at ordinary temperatures is given at 1.99×10^4 to 5.35×10^4 per cubic centimeter; or the resistance varies from 5.02×10^{-8} to 1.87×10^{-8} ohms.

SHEET-IRON ANODES.

As already remarked, logically sheet-iron anodes should be classified under "Soluble anodes," but it is more convenient to discuss

* Landolt, H. H., Landolt-Börnstein Physikallisch-chemische Tabellen, 3d ed., 1905, p. 723.

them here. Sheet-iron anodes were used first by Siemens. It is well known that with a weak current the iron combines with the cyanide and cyanogen to form potassium ferrocyanide, which forms, with various iron oxides, Prussian blue and other compounds. If the current is of a certain density a protecting oxide coating forms on the anode, from the oxygen set free there, and the anode lasts very well. But if there are chlorides in the solution, the iron salt of this acid forms at the anodes and is secondarily decomposed by the alkaline solution, forming at last hard, warty crusts of ferric hydrate several inches thick. Sulphates seem to be nearly as harmless, but chlorides which are harmless to carbon are fatal to iron anodes.

These crusts increase the electrical resistance. Moreover, they contain considerable cyanide of gold, which has traveled to the anode and become entangled, making an objectionable residue that is hard to treat. Also, the cost of the quantity of iron thus destroyed, amounting in large-scale work to many tons, is considerable.

"PASSIVE IRON" ANODES.

The well-known but elusive substance known as "passive iron" was first suggested as an anode in cyanide solutions by Hittorf,^a and it has been again proposed by Kern^b as a cure for the trouble above described. For the purpose he treated the metal as follows:

The iron anodes were rendered passive by first "removing grease by digesting them in hot caustic soda solution, rinsing in water, then putting them into a dilute solution of mixed sulphuric and hydrochloric acids in order to remove all oxide, washing with clean water, thoroughly drying, and finally placing them in concentrated nitric acid and allowing them to remain for about two hours. By this treatment the iron plates were coated with a film of magnetic oxide (Fe_3O_4). On removing the plates from the concentrated nitric acid they were immediately washed with clean water and thoroughly dried to prevent the formation of ferric oxide (Fe_2O_3)." The manner of drying is not mentioned. It is an important omission in such a delicate operation. Kern expresses great satisfaction with the "passive iron anodes," but the experiments he describes only lasted a couple of hours, and he does not seem to have tried the effect of NaCl and Na_2SO_4 . Long experience shows that the conditions governing "passive iron" are too uncertain and too little understood to pin much faith to anodes so constructed. I have been unable to find any of the many directions for producing them to give a result that would make a permanent anode that would stand up with

^aHittorf, W., Über passivität der Metalle: *Ztschr. Phys. Chemie*, Bd. 24, 1900, p. 395.

^bKern, E. F., Electrolysis of cyanide solutions: *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, pp. 241-270.

cyanide solutions containing chlorides under tensions of three or four volts and not break down. Ferric hydrate and Prussian blue soon form the usual crusts and the anode fails utterly.

If some such method could be perfected it would be of the greatest practical utility, not only in electrolysis, but in all the other industrial uses of iron. The investigation of "passive iron" has occupied the attention of many able men for years, but there is no general agreement as to the cause of the phenomena. I use the plural advisedly. If the cause were once surely settled some practical methods would almost certainly result. But "where doctors disagree who shall decide?" The patient usually succumbs, and that is what happens to the "passive iron" anode when it is put to practical use.

An interesting symposium on the passivity of metals was held November 12, 1913, in London by the Faraday Society.^a The discussion, which is well worth close study, shows that the phenomena are too varied to be covered by a single theory. The three important theories that are held are the following:

1. The oxide film theory, proposed by Faraday.^b

2. The valency theory of Finkelstein and Krüger^c and of Müller.^d This assumes that the metal is in an electric condition which alters its valence and renders it more "noble." Finkelstein assumes that in the case of ordinary iron both bivalent and trivalent atoms are present; that the passivity consists in either removing the bivalent atoms from the surface or changing them into the trivalent kind. Hittorf^e disagrees with Faraday in the assumption, and although he seems to incline to some such view, does not commit himself.

3. The reaction velocity theory of Le Blanc, which presupposes two possible conflicting reactions, the changing rate of which produces the various effects. There are three or four modifications of this theory which are rather metaphysical in their refinements and to discuss them in detail would take us too far afield.

As early as July, 1900, I had hopes of utilizing "passive iron" anodes. Iron was made "passive" in a mixture of strong nitric and sulphuric acids, and after washing the specimens of iron so treated were made anodes in potassium cyanide to which a little silver nitrate had been added. The anodes were attacked with free formation of

^a Senter, G., and others, The passivity of metals: Trans. Faraday Soc., vol. 9, March, 1914, pp. 203-290; abstracted in Met. and Chem. Eng., vol. 11, December, 1913, p. 679.

^b Faraday, Michael, On a peculiar voltaic condition of iron: Philos. Mag., vol. 9, 1836, pp. 57-65.

^c Finkelstein, Alexis, Über passives Eisen: Ztschr. Phys. Chemie, Bd. 39, 1902, pp. 91-110.

^d Müller, W. J., Zur Passivität der Metalle: Ztschr. Phys. Chemie, Bd. 48, 1904, pp. 577-584.

^e Hittorf, W., Über die Passivität der Metalle: Ztschr. Phys. Chemie, Bd. 34, 1900, p. 393.

hydrates and other compounds. Iron boiled in paraffin was similarly attacked. Electric-light carbon which had been boiled in paraffin to fill the pores was at eight volts disintegrated on the surface into a sludge.

On July 7, 1900, I conducted an experiment with a 0.5 per cent cyanide solution containing gold and silver as cyanides and 1.5 per cent common salt. For the anode I used a small ornamental cast-iron scroll that had been well coated with a magnetic iron coating by the Bower-Barff process. Four volts gave a current of 0.100 ampere, with a wetted area of about 10 square inches. After $3\frac{1}{2}$ hours the current had risen to 0.250 ampere. Red hydrate of iron, but no Prussian blue, had formed on each of the little spots where the protective covering had scaled off. The main body of the coating did not appear to have been attacked in the least. The scroll has since lain around the laboratory and been exposed to various atmospheric conditions without further change, which goes to show how energetic anodic oxidation is, particularly when assisted by a little salt.

Two days after making the test above described I took a wire nail about one-eighth inch thick and heated it to redness till it acquired a uniform coat of scale. When cold this proved to be a fairly good conductor. I made the nail an anode in a bath of 200 c. c. of 0.5 per cent KCy solution, containing $KAgCy_2$, to which 20 c. c. of a saturated solution of NaCl had been added. The nail was immersed about 3 inches. At 2 volts it permitted the passage of a current of 0.001 to 0.002 ampere, and was very little acted on except at the point. I allowed the current to run over night and found in the morning that it had risen to 0.100 ampere. A large quantity of a greenish-yellow slime had formed and settled out. The nail was half eaten through at the end and near the middle, but otherwise the black magnetic coating was intact. When the slimy hydrates were removed, the part eaten out below the slime was bright and white like silver. Evidently where the coating was intact it protected the iron from attack, but wherever there was a defect in the coating the iron was corroded rapidly, evidently forming a short-circuit couple between the iron and the oxide coating, which had made the corrosion more active on account of the "local action." I have experimented in this direction and have had hopes of working out a method that will give a firm coherent coating of magnetite on cylindrical iron rods, enabling the construction of a grid anode coated with magnetic oxide of iron similar to the grid anode of peroxide of lead.

Evidently in the two tests cited the "passivity" was due to an oxide film, and the explanation suggested by Faraday holds good. Films so formed are not necessarily of Fe_3O_4 , as assumed by Prof. Kern, but seem to be mixtures of FeO and Fe_2O_3 in varying propor-

tions. There is more of the Fe_2O_3 on the outside and more FeO on the inside. The films are usually magnetic. The least defect in the rather brittle coating exposes the interior and corrosion ensues in a way similar to that of iron poorly electroplated with copper. The iron rusts rapidly, even under the copper coating, and scales worse than does the magnetic coating. If the magnetic coating is thick, it chips very easily, but if it is thin it seems to be porous. These defects are recognized by the Bower-Barff concern, which always dips thin light-coated grill work for interiors in melted paraffin to fill the pores in the coating. Whether a durable coating of this oxide can be obtained for anodes subject to bending and warping in handling is open to doubt.

The failure of iron anodes made "passive" in strong nitric acids, or by anodic polarization in alkaline oxidizing agents, may be due also to porosity. The film, or whatever the protecting condition is, appears to be too delicate for practical use. Grave^a claims that the pure iron is passive but the presence of hydrogen ions destroys this passivity. The "passive" state created by acid treatments is evidently too unstable to serve as the base of any industrial method in the present state of knowledge, but the importance of the subject warrants further research.

ANODES OF MAGNETITE.

Henry Blackman (U. S. patent 568231, Sept. 22, 1896) claims the use of electrically conductive anodes of iron oxide in a dense impermeable mass. He specifies the use of magnetite or ilmenite cut from a slab of the mineral, which would be a difficult task. He specifies also the agglomeration of grains of these minerals by tar or sugar or by means of suitable fluxes. This, of course, is objectionable as the binder yields and also acts as an insulating agent.

I have tried a pervious mass of loose magnetite grains held in a box with cheesecloth sides, but the mineral in this shape is neither sufficiently pervious to the solution nor of sufficient conductivity.

Heinrich Specketer (U. S. patent 931513, Aug. 7, 1909) claims the use of anodes cast in rods from molten magnetite. He fuses Fe_2O_3 , which changes on melting into Fe_3O_4 with a slight excess of FeO , the latter crystallizing in a different system from the magnetite and causing it to disintegrate. This oxide is removed by adding a small amount of finely pulverized Fe_2O_3 to the melt just before pouring into the molds. The mass solidifies on the outside first and the molten interior can be poured out, leaving a hollow tube which can be electroplated with metal on the inside, to make it a better con-

^a Grave, Ernst, Neue Untersuchungen über die Passivität von Metallen: Ztschr. Phys. Chem., Bd. 77, 1911, pp. 513-576.

ductor, or a rod of iron, copper, or nickel can be inserted in the still plastic interior. This patent has been assigned to the Chemische Fabrik Griesheim Electron, of Frankfort, Germany. The firm has undertaken its manufacture.

A part of one of these anodes, broken from a larger part, was sent me by Mr. Haigh, president of the Moore Filter Co. It was formerly used in connection with the Clancy process. The piece is of a uniform, bluish-black color, is about 10 inches long by five-eighths inch in diameter, and is cylindrical. An inch and a half near the top has been electroplated with copper to facilitate making contact. The coating is thin and has partly scaled off.

I have used this specimen as an anode in a 0.2 per cent cyanide solution without sodium chloride and sodium sulphate, with 1 per cent sodium chloride in one experiment, and with 1 per cent sodium sulphate in another, employing 4 volts, without the slightest apparent formation of either ferrous hydrate or Prussian blue, and without any change of appearance either of the anode or the solution. I did not use the anode for any great length of time, but it appears to be very durable. The outer cast surface is a deep bluish black, almost like velvet, but the fractured surface is rather bright and crystalline, like fractured magnetite. The material has the disadvantage of being brittle, but is much less so than Acheson graphite. The magnetic anode is not as good a conductor as the graphite, but is, on the whole, one of the most promising anodes that I have seen. I am informed that the material is not so permanent with an alternating current, and also that it has been abandoned as an anode for the Clancy process, being found less satisfactory than Acheson's "extruded graphite." Such graphite when soaked in paraffin is said to give better results with that process.

The permanence of the magnetite anode suggests that, in this case at least, the oxide film, as suggested by Faraday, is the true cause of the passivity. In a thin film, formed in the wet way, porosities and defects of the film may be the cause of the unreliability and frequent failures. When the film becomes ruptured, exposing only the minutest spot, an intense local action begins and destruction follows. It is possible that electrolytic lining of the hollow magnetic anode with copper in a similar way, or filling it with a rod of iron or copper while still hot, might also cause shrinkage cracks in the film and similarly result in failure.

A serious defect of the magnetite anode is that it is not a very good conductor. Landolt and Bernstein* give for Swedish magnetite a resistance of 0.595 ohm per cubic centimeter, conductivity 1.68. I have found a much less resistance by testing the anode rod just de-

* Landolt, H. H., Landolt-Börnstein Physikallisch-chemische Tabellen, 3d ed., 1905, p. 724.

scribed, a result which I believe to be nearer the true value. The mean of four closely agreeing tests conducted at room temperature was 0.0319 ohm per cubic centimeter (or a conductivity of 31.4). This makes the resistance nearly 20,000 times as great as that of copper, more than 3,000 times as great as that of iron, and more than 10 times as great as that of a good electric-light carbon. This is a serious drawback. Of course if the metallic core could be used, without any failure in the protecting surface, this resistance would be much cut down. Further details from the manufacturers have not come to hand.

CATHODES.

Satisfactory cathodes, as such, are not so difficult to obtain. Many suitable substances are available. But when, as in the Christy electroconcentrating process, a cathode is desired that may serve alternately as a cathode and as an anode, the problem is not so easy.

SOLUBLE CATHODES.

Mercury and amalgamated copper cathodes are specified in numerous patented processes, but serious chemical and physical objections to using these materials have been sufficiently explained. An additional objection is the cost of the large amount of these expensive metals that must be used to obtain effective deposition from the dilute solutions required in practice. Except in certain exceptional electrolytic methods where strong solutions are used and where regeneration of the cyanide and the caustic alkali is possible, it may be said that the use of quicksilver and copper is out of the question.

SHEET ALUMINUM.

The use of sheet aluminum has been proposed by Cowper-Coles,* who claims that the cathode should fulfill the following conditions:

1. The gold should be adherent during the process of deposition.
2. The gold should be capable of being readily stripped, after removal from the electrolyzing cell.
3. The cathode should be electropositive to the gold in solution, to insure its being coated with gold on immersion.

It is affirmed that deposited gold is easily stripped from the cathodes after deposition, and claimed that the metal is recovered successfully from a solution containing 2.5 pennyweight of gold per ton and 0.01 per cent KCy. This solution is treated for 10 hours, with a flow of 15 gallons per 100 hours for every cubic foot of the deposition box, and for 3 square feet of cathode surface. Six volts

* Cowper-Coles. Sherard, Some notes on the recovery of gold from cyanide solutions: Trans. Inst. Min. and Met., vol. 6, 1897-8, p. 219.

are mentioned as being used. However, there is great danger of the cathodes decomposing the water and becoming coated with aluminic hydrate in spite of the current, and this of course would put an end to their usefulness.

The idea of using a metal electropositive to gold to prevent resolution of the gold in case the current fails is an ingenious one, and has been proposed by Bettal and Andreoli, both of whom use zinc for the purpose. It must be remembered that the zinc or aluminum can act as an electropositive metal and protect gold on a failure of the current only when the gold is deposited so loosely as to allow the solution access to the zinc or aluminum to form a couple. If the gold is deposited as a coherent sheet, as proposed by Cowper-Coles,^a no such protective action is possible. The anode and the gold-coated cathode then form the couple, and the gold dissolves if the current fails, in spite of the protective aluminum or zinc. Also, when mercury is in solution it amalgamates with the aluminum of the cathodes and causes the latter to heat, oxidize, and disintegrate in a most remarkable manner. In my judgment, these objections suffice to invalidate the use of aluminum for cathodes in cyanide solution.

INSOLUBLE CATHODES.

HARD CARBON AND GRAPHITE.

Hard carbon and graphite can be used in the form of sheets, both as cathodes and afterwards as anodes, in a stripping solution of pure cyanide at a low voltage without any difficulty whatever. When the gold has been concentrated on Acheson graphite sheet cathodes it may be readily stripped for the mint. It does not even need to be melted down.

There are, however, four objections to such electrodes: (1) Sheet electrodes do not offer sufficient surface to permit of rapid precipitation from dilute solutions. See the comparison of results in using sheet-iron and iron wire-cloth cathodes (p. 110). (2) Graphite and carbon in other ordinary forms are brittle, and electrodes of these materials are easily broken if much handled. (3) On account of the friable character of the material it is not possible to design a compact deposition box, as the cathode sheets must be at least one-fourth to three-eighths inch or more thick, whereas with wire cloth five times the precipitating surface may be provided in the same space. (4) The electrodes are costly. However, notwithstanding all these drawbacks it must be said that such electrodes furnish a physically workable solution of the difficulty.

^a Cowper-Coles. Sherard, Some notes on the recovery of gold from cyanide solutions: *Trans. Inst. Min. and Met.*, vol. 6, 1897-8, p. 219.

SHEET IRON.

Gold comes down very well on a surface of sheet iron; but, as already pointed out, the defect of plane sheets is the lack of sufficient precipitating surface. I have not found it necessary to scale the sheet iron by pickling; the electrolytic current soon precipitates enough gold and silver to cover the magnetic oxide scale, and in time the action even will cure rusted surfaces. It is, of course, an added expense to subject electrodes to pickling, and the cost of this would be considerable. The method of pickling suggested by Julian and Smart^a is as follows:

We have also used iron with success (as cathodes), and we find that in order to get the best results all the oxides should be removed from its surface. This we have done by immersing the plates in the following mixture: 100 parts of water, 10 sulphuric in which 1.5 of zinc is dissolved, and then adding 10 of nitric acid. If then washed in an abundance of water and kept in an alkaline solution they remain nearly as bright as silver. In this state they receive a perfectly uniform deposit. Lead can not be used to remove the deposit, as this injures the iron surface for reuse, but if placed in narrow iron boxes containing 2 to 5 per cent KCy solution and connected with Daniell cells in parallel to form the anode, while the box is the cathode, the gold and silver rapidly dissolve off without reprecipitating, leaving the plate perfectly bright and clean and ready to go back into the precipitating box.

The solution is then evaporated to dryness and melted to get the gold.

This method would be more expensive than ordinary pickling, but I can testify that it gives good results. The plates remain bright in alkaline solutions almost indefinitely. Even when used as anodes, with a voltage below 1, such plates stand up fairly well, but at higher voltages they oxidize as badly as ordinary iron. The metal can hardly be said to be "passive" when it acts as a cathode. It is possible that a film of zinc forms on its surface, although this seems improbable in the presence of nitric and sulphuric acids, or the film may be nitride of iron. At any rate the surface seems to be more permanent than "pickled" iron surfaces usually are.

As I have already pointed out, the intermittent nature of the experiments introduces a difficulty that would not occur to the same extent in continuous operation. At the end of a run the iron wire-cloth cathodes with gold or silver deposited on them had to be removed from the cyanide solution. It was not feasible to dry them by artificial heat, because the uneven expansion of the metals might cause scaling. If allowed to dry in the air at room temperatures the dampness of the wires soon caused local action, gold and silver were attacked slightly and dissolved in the cyanide solution with which the wires were wet, and after the cyanide was exhausted the

^a Julian, H. F., and Smart, Edgar, Cyaniding gold and silver ores, 1904, p. 140.

iron began to rust by local action under the coating. This trouble appeared when the cathodes were again returned to the deposition box, and, as has been seen, during the first hour or more the gold and silver dissolved in spite of the current. Not until the rust had been reduced was the gold again precipitated. In continuous work the cathodes would remain in the deposition box till they were to go into the stripping box, and when stripped they would go back into the deposition box. The difficulty would not then appear to the same extent. However, it was highly desirable to cure the evil altogether.

Many devices were resorted to. Among others the cathodes were immersed in gasoline or refined petroleum as soon as removed from the deposition box. As these liquids contained no oxygen, it was hoped to preserve the cathodes from rusting. They have such an effect, except at some points where occluded oxygenated water remained and acted. But a worse evil arose. The gasoline, like all petroleum products commercially applicable, contained so much dissolved heavy hydrocarbon that when the cathodes were removed for use an insulating residue was left on the entire surface, rendering them almost nonconductors. It is possible that the cathodes might have been preserved if they had been dried in vacuum and kept in a vacuum. But this did not appear to be practicable.

Heating the iron-wire gauze to redness in an atmosphere of ammonia gas was tried. Hydrogen was set free and nitride of iron of a silver-gray color formed, and the cathodes no longer rusted in any ordinary atmosphere. A piece of such gauze so treated has been exposed 10 years to the atmosphere of the laboratory without perceptible change. But the treatment had made the wire brittle as a pipestem, and when used as an anode under a strong current it failed as before. Possibly such a treatment might be used if it were not pushed too far, but the extreme brittleness resulting led me to drop the method.

PLATINUM.

Cathodes of platinum-wire gauze of 16 meshes to the linear inch, of 0.5-mm. wire, united like those of iron-wire cloth into "bunches of five," would be satisfactory as regards both rapid precipitation and perfect stripping. Some method of rapid drying would still be necessary to avoid resolution of the deposited metal in intermittent use, but the undercutting by rusting would have been avoided. However, calculation soon showed that the cost of such cathodes was prohibitory, and I was forced to compromise. I ordered some wire-cloth cathodes of 16 mesh (linear) to be made by Hereaus in Germany. The metal used was platinum with 10 per cent iridium, but the wires were only 0.1 mm. in diameter and the

conducting wires welded to the edge of the cloth were 0.5 mm. thick. The cathodes were made 11 by 11 cm. in size, with an ebonite frame at the edges to give the required stiffness. The section exposed to the solution was 10 by 10 cm. The 88 electrodes cost, at the price then prevailing, in 1902, nearly \$250. Seventy-two of these anodes and cathodes, with insulators, could be compacted into an 8-inch length of box with an effective section 4 by 4 inches, or into 128 cubic inches, or 0.074 cubic foot. That would have made a total cost for electrodes of \$2,900 a cubic foot. At the rate of even 1,000 cubic feet of solution treated a day in such a box, or 31 tons a day, or 11,315 tons a year, the interest on the cost of the platinum would be \$145 a year, or \$0.0128 per ton of solution precipitated. At the present prices of platinum this item would be far greater. The cost alone would be excessive; but the platinum cathodes at the start had only one-fifth the precipitating surface of the iron ones, and the precipitation was at only about half the rate at best. As they became coated with gold and silver, however, the area would increase, and the precipitating rate would increase correspondingly until the gauze would be choked with deposited metal.

The use of iron-wire gauze electroplated with platinum, or composite wires with an iron core and a platinum face drawn down through draw plates, might be feasible. But I had doubt as to whether a layer of platinum as thin as it would be economical to employ would be impervious to the OH ions. If the platinum were not impervious, the electrode would soon be destroyed. The ends of the composite drawn wires would also be a weak point for attack.

The most favorable ratio between wire diameter and distance between wires to give maximum depositing area with minimum resistance to flow of solution is when the diameter of the wire is one-third the distance between the wires, center to center, as can easily be shown mathematically. If this ratio is maintained, the total wire surface (warp and woof together) is a little more than that of both sides of a solid sheet of metal. Diameters of wire between 0.5 and 1 mm. give the best results.

IRON-WIRE CLOTH.

The many advantages of iron-wire cloth cathodes, such as the large precipitating area, the dense current that they will endure, their great permeability, and the compactness of design which they lead to, have made me very reluctant to abandon them. As before explained, the oxidation by "local action" between experiments followed by subsequent rusting and resolution would not occur in continuous use. Very little oxidation occurs in the stripping box if the stripping current is kept below 1 volt. In order to prevent leaving the electrodes too

long in the stripping box, which would be wasteful of current and would be attended with other bad effects as is claimed by Uslar,^a they could be suspended in the box by using a device like Roseleur's plating balance described by McMillan.^b This could be made to break the stripping circuit automatically and sound an alarm if thought necessary. Anodes of carbon or of melted magnetite could be used when desired to recover the dissolved gold in sheets at a slightly higher voltage from the clean-up solution. As the stripped cathode that has served as the anode of the clean-up box would be wet with a rich solution, and would have suffered an incipient oxidation, it would be inserted first as cathode for a few minutes in the so-called "hospital cell." This cell would contain 1 or 2 per cent caustic potash or soda, and with 4 or 5 volts a strong current giving a violent evolution of hydrogen would soon bring back any oxidized spots into such a state that they would be at once effective when again inserted in the deposition box.

When iron-wire cloth cathodes have become badly rusted they are easily brought into normal condition by heating in a closed iron box to about 400° C. (below red heat) in an atmosphere of hydrogen, carbon monoxide, or vapors of gasoline. Care must, of course, be taken to displace the air from such receptacle by the reducing vapor before applying the heat. The excess vapor used for the reduction would be burned, outside the box, in a suitable burner to furnish the heat necessary for the reaction. The cathodes, when badly rusted, may be brought back to duty by immersing them in powdered charcoal and heating them to a temperature below redness. Care should be taken not to overheat and melt them.

1. PERVIOUS CATHODES OF BROKEN COKE, CARBON, OR GRAPHITE.

In order to escape the difficulties with the iron-wire cloth cathodes I have tried also pervious cathodes composed of $\frac{1}{4}$ -inch fragments of hard coke, as well as retort carbon and graphite. Cathodes of these materials were formed by placing the material in boxes having cheesecloth sides, so that the solution could flow through the mass. On April 5, 1904, I was granted United States patent No. 756328 for a process of recovery of gold and silver from cyanide solutions along these lines. Figures 30 and 31 taken from the patent specifications will serve to illustrate the devices used.

The general process is shown in diagram 1, figure 30. *B* is the storage tank for the solution; *A*, the deposition box or electrochemical cell. The figures shows the anodes *d* (marked +) and the

^a Von Uslar, Manuel, and Eriwein, Georg, *Cyanid-Prozesse sur Goldgewinnung*: Monograph. angew. Elektrochemie, Halle, Germany, 1903, 100 pp.

^b McMillan, W. G., *Treatise on electro-metallurgy*, 3d ed., 1910, p. 101.

cathodes e , marked with a minus (—) sign. These are connected in parallel with a shunt-wound or separately excited dynamo D , provided with a suitable ammeter F and a voltmeter C . The solution

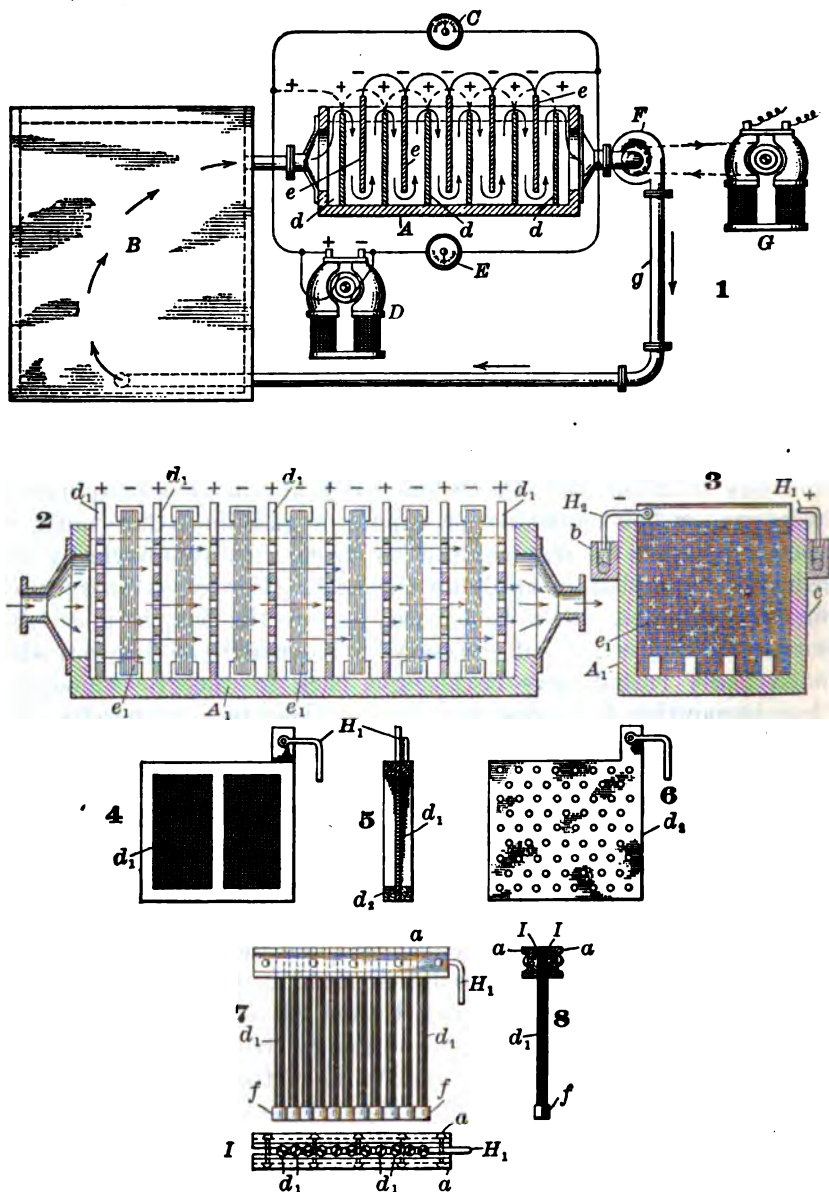


FIGURE 30.—Apparatus for recovery of gold and silver from cyanide solutions.

flows through the deposition box A from the tank B through a suitable centrifugal pump F , driven by a motor G , and back again through pipe g to the tank, and so on, rapidly, continuously, and

repeatedly and in such a manner that the solution is brought into intimate contact with anodes and cathodes in rapid alternation until

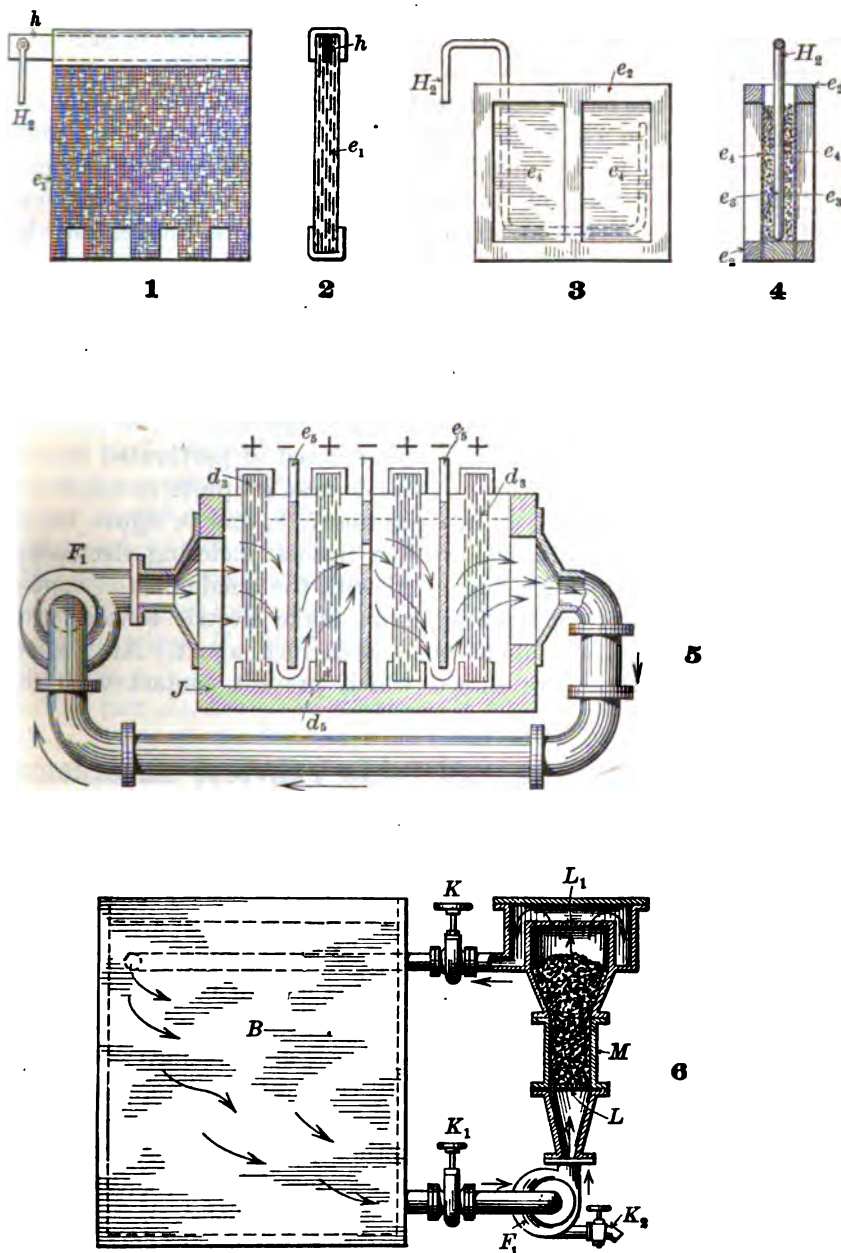


FIGURE 31.—Details of electrodes and of deposition box. Lettering similar to fig. 30.

the gold and silver content has been sufficiently reduced, when the solution is discharged and a new lot treated.

Diagram 1 of figure 30 shows also the common form of deposition-box A , with removable cathodes c . The solution flows in a zigzag course through the box, as is indicated by the arrows; but the form of deposition box or electrochemical cell that I prefer is one with pervious electrodes, as shown in diagrams 2 and 3, figure 30. In these diagrams the box is designated by A_1 , the anodes, d_1 , are marked with the plus sign (+). The anodes are made of any electroconducting substance not too much acted on by the solution, such as iron, lead, peroxide of lead, platinum, or, for pressures not greatly exceeding 1.5 volts, dense carbon or graphite. These anodes may be constructed of wires, rods, gauze, or perforated sheets, in any form or manner to be pervious to the solution. The wire-gauze d is best mounted to prevent contact with the cathodes, in a parafined wooden frame d_2 , as in diagrams 4 and 5, figure 30, connected electrically with an amalgamated copper wire H_1 . The perforated sheet-metal anodes d_2 are constructed as in diagram 6, figure 30. The graphite or carbon anodes may be similarly constructed of perforated blocks; but I prefer to construct them of the densest graphite or electrode-carbon rods d_1 , mounted, as in diagrams 7, 8, and 9, figure 30, in which a represent parafined wooden frames, inclosing electroconducting strips I , which are connected with the amalgamated copper wire H_1 . The electroconducting strips make electric contact with the carbon rods placed about one-eighth inch apart. At f short strips of rubber hose are shown which prevent contact with the cathodes.

METHOD OF PREPARING AND USING PERVIOUS ELECTRODES.

When wire-cloth cathodes are used the wire cloth is cut to a size to fit the box, and the sheets c_1 are clamped together, as in diagrams 1 and 2, figure 31, in bunches of five or ten sheets, electrically connected with a sheet-iron strip h , to which is riveted an amalgamated copper wire H_2 . Before the sheets of wire gauze are clamped together they should be preferably heated to dull redness for about one minute to burn the oil usually left on the surface by the manufacturer and to coat them with a thin coating of magnetic oxide of iron. They should not be exposed to the heat too long or a thick scale forms that easily cracks off.

Instead of iron-wire gauze any other pervious conducting substance may be used as a cathode, such as metallic filaments or cloth saturated with precipitated silver, gold, or other metal, thus exposing a large surface. A metal surface of this kind may be prepared by inserting a sheet of metallic wire cloth (of iron, for instance) as a basis for a cathode, into a suitable electrochemical cell provided with pervious anodes. Through this cell is circulated a strong solu-

tion of gold, silver, copper, lead, or other electronegative metal, during the passage of a dense electric current, to precipitate such metal on the surface of the wire cloth in a pervious or spongy layer. This forms an electrolytic filter of enormous cathode area in a small compass. The anodes and cathodes thus formed may be arranged as shown in diagram 2, figure 30, or they may be arranged horizontally one above the other and the dilute cyanide solutions from which the gold and silver are to be removed may be forced to circulate rapidly through such pervious anodes and cathodes.

Also, for the purpose described, granulated metal, pulverized coke, or electrocarbon dust e_1 , properly sized, may be placed in a wooden framework e_2 , covered with cheesecloth e_3 , and electrically connected by an iron wire ending in an amalgamated copper wire H_2 , as shown in elevation in diagram 3, figure 31, and in cross section in diagram 4, figure 31. The electrodes are separated one-half to one-fourth inch to prevent short-circuiting. In diagrams 2 and 3, figure 30, is shown the arrangement of such a deposition box with pervious anodes and removable pervious cathodes. Electric connections in parallel are made with the anodes and cathodes, as shown in the sectional view 3, figure 30, by the box b , connected with the negative pole, and by the box c , connected with the positive pole of the dynamo. The wooden boxes b and c contain omnibus bars of amalgamated copper and connecting holes filled with mercury, into which dip the wires H_1 from the anodes and the wires H_2 from the cathodes.

The box A_1 , shown in diagrams 2 and 3, figure 30, is placed in the position shown for box A in diagram 1, figure 30, being substituted for that box, and the solution is forced to circulate repeatedly from tank B through the box A_1 and through the pervious anodes and cathodes, and back again, so as to be brought into intimate contact with anodes and cathodes in rapid alternation until its gold and silver content is sufficiently reduced. When the voltage is kept constant at any point (at two volts, for instance) the completion of the treatment is indicated by the number of amperes of the electric current. The number is high at first, but suddenly drops and becomes constant when the gold and silver content of the solution is exhausted. Since the fall of electric current could be caused by an accidental increase of resistance, it is necessary always to assay the solution before discharging the lot under treatment.

I prefer to concentrate the gold and silver thus deposited on removable cathodes in the manner shown in my former patent, No. 643096, February 6, 1900. However, I have been able to increase the precipitating capacity of the deposition box by the new method described here to far exceed the capacity of the clean-up box there shown. I have also been able to apply the same principle of rapid

circulation to that box, and thus to increase its capacity. Diagram 5, figure 31, shows how this is effected. A centrifugal or other pump F_1 forces the cyanide solution, which should be as free as possible from chlorides and sulphates, to circulate rapidly through the clean-up box J , preferably, though not necessarily, at such a rate as to prevent the evolution of gas at the cathodes. These consist, preferably, of sheets of iron e_6 , marked with a minus (—) sign in the figure and arranged, as shown, to force the solution to pass alternately up and down through the box and at the same time through the pervious secondary anodes d_3 , marked with a plus (+) sign. These secondary anodes are the gold and silver laden primary cathodes e_1 , from the deposition box, shown in diagrams 2 and 3, figure 30. Although pervious cathodes e_1 to collect the silver and gold in the deposition box are preferable, impervious cathodes, e , shown in diagram 1, figure 30, can be used. The gold and silver content is then concentrated on the secondary cathodes in a similar manner. The secondary cathodes e_6 , of sheet iron or other metal, marked minus (—) are, before being inserted in the clean-up box, coated with a very thin layer of electroplater's graphite and vaseline. With an electromotive force of about one-half volt, the gold and silver are rapidly removed from the removable secondary anodes d_3 (primary cathodes e_1 in the deposition box) and deposited on the secondary cathodes e_6 , as solid sheets. These sheets may be pulled off at convenient intervals and are ready for the mint without further treatment. I have thus produced solid sheets of bullion one-eighth of an inch thick and 996 fine. If an extremely dense electric current is used in the clean-up box, the gold and silver will be deposited on the secondary cathodes in the form of a loose powder, which is easily removed with a stiff brush and can be melted down into bars. The rapid circulation of the solution permits the concentration of the metal with a dense current, and greater rapidity is insured in a small box.

Also, it is evident that, instead of having two separate boxes—a deposition box and a clean-up box—the same result may be obtained with a deposition box that serves both purposes in alternation. This is particularly the case when pervious cathodes of broken carbon or graphite are used, for these are bulky and inconvenient to remove to a clean-up box. Instead of doing this, when the cathodes become sufficiently charged with gold and silver in the deposition box after the circulation of a large volume of cyanide and ore solution, the circulation from the storage tank is cut off. Then the same or a fresh cyanide solution is made to circulate rapidly and repeatedly through the box (see diagram 2, fig. 30) by means of a pump, as in diagram 5, figure 31, except that now the primary cathodes of diagram 2, figure 30, marked minus (—), are connected with the plus

(+) pole of the dynamo, to become secondary anodes, while the original anodes (diagram 2, fig. 30), marked plus (+), are replaced by secondary cathodes of sheet iron which have been previously coated with a mixture of graphite and vaseline. The secondary cathodes on being connected with the negative (—) pole of the dynamo become coated with the gold in a concentrated form, which had been previously distributed on a very large primary cathode area. The secondary cathodes may be of the form of those marked minus (—) in diagram 5, figure 31, or they may be as shown in diagram 6, figure 30, or they may be of separate strips of iron or other conducting material, arranged like the slats of a window blind, so as to be pervious to the solution. It is preferable to circulate the solution at such speed that evolution of hydrogen gas is practically avoided at the cathode; if, owing to the density of the electric current used, this is not possible the velocity should not be less than 1 foot per minute. When the gold content of the original cathodes, now secondary anodes, has been recovered, the secondary cathodes, now laden with concentrated gold, are removed, the original anodes are replaced, the electric connections changed to the original condition, and the stock solution turned again through the tank, which now serves as a deposition box, as shown in diagrams 1 and 2, figure 30.

The same principle of rapid circulation and repeated treatment in rapid alternation of a large volume of solution in a small deposition box containing pervious electrodes may also be applied to the electrochemical recovery of gold and silver from cyanide solutions without the use of a dynamo or external source of electricity, the electric energy being furnished by such an electropositive metal, as zinc, in the form of fine granules, shavings, or dust. This system is shown in diagram 6, figure 31. The figure shows at *B*, the large storage tank containing the solution to be treated, a centrifugal or other pump *F*₁, and a vessel *M*, containing zinc in the form of shavings or other form, exposing a large surface. The zinc rests on a filter *L*. A filter *L*₁ may be used at the top of the box to avoid the loss of fine particles, if desired. *K*, *K*₁, and *K*₂ represent valves to control the flow. The solution is forced to circulate from the tank *B* through the zinc and back again repeatedly (preferably, though not necessarily) at such a rate as to substantially prevent the evolution of hydrogen. The gold and silver are rapidly precipitated by the electrolytic action set up between the solution and the zinc. The zinc particles act as anodes and the gold and silver particles, which are at once deposited in a porous or pervious state upon the zinc, act as cathodes. The zinc anodes covered with this pervious coating of gold and silver form a combination of pervious anodes and cathodes, and when the solution is forced to circulate rapidly and repeatedly through them it is brought into intimate contact with anodes and

cathodes in rapid alternation. Although the electromotive force of the combination is small, the current is short-circuited, and the resistance therefore is smaller than in any other form of cell, and hence the gold and silver are rapidly precipitated. In this type of process it is important that the gold and silver be precipitated on the zinc in a loosely adherent porous film in order that the zinc and the precipitated metal form a set of electrodes pervious to the solution. In some instances the precious metal forms a solid impervious film on the surface of the zinc. When this happens, precipitation ceases. In such an event the zinc, before use, should be coated by dipping it into a dilute solution of nitrate of silver or acetate of lead, forming a pervious film of silver or lead. When the zinc is sufficiently fine the precaution is unnecessary.

By the ordinary zinc process, owing to local action, 5 to 20 ounces of zinc are required to precipitate 1 ounce of silver or gold, but with the Christy process practically complete precipitation of 1 ounce of silver with 0.57 ounce zinc has been effected, and results nearly as good with gold. In practice the use of about 2 ounces of zinc to 1 of gold and silver is preferable, but even then the use of the process will save zinc and reduce the cost of refining the precipitate.

By the application of the same principle the precipitate can be refined without the use of acid by circulating rapidly and repeatedly a solution of cyanide, nitrate, or sulphate of silver, of cyanide or chloride of gold, or soluble salt of copper, through the precipitate, so that such solution may come into contact, in rapid alternation, with the residual particles of zinc, acting as pervious anodes, and of deposited gold and silver, acting as pervious cathodes, until the residual zinc is dissolved. After refining the precipitate in this manner the silver or gold, left in the refining solution, is recovered by the use of fresh zinc, as before.

Most of my work, though not all, was done with pure solutions, and desiring to see how the process would work with stock solutions from the treatment of ores, I obtained in 1905, through the kindness of Mr. Charles Rhodes, manager of the Waihi mine, New Zealand, and of Mr. W. W. Mein, general manager of Robinson mine, Johannesburg, an opportunity to have tests made at these places.

I much appreciate the trouble the experimenters took with these tests, but none of the experiments were carried out exactly as I directed. In some tests the lack of facilities at hand made this impossible; in others the ideas of the experimenter led to changes in details without realization of the detrimental effect. Most serious of all was the failure to record all the necessary variables. Thus the causes of success or failure in many of the experiments can not be determined. Circulation by means of the pump, which should have been easily controlled, was found troublesome and results in

some tests seem to indicate that the boxes did not get the proper treatment. I present here the results obtained by other men in tests conducted at my request as described.

EXPERIMENTS AT WAIHI MINE, NEW ZEALAND.

The experiments at Waihi mine, New Zealand, were conducted by Herbert W. Hopkins, metallurgist, Victoria mill, Waikino, in 1905.

The solution employed for these tests was that which had been used in treating a clean gold and silver bearing quartz ore containing 1 per cent pyrite. The solution was heavily charged with lime to settle the slimes. The box used was 3 feet 4 inches by 1 foot by 10 inches. The electrodes were immersed to present an area 11 by 9 inches, which later was increased to 11 by 10 inches.

EXPERIMENTS WITH FLAT OR PLANE ELECTRODES.

SERIES A TESTS.

Both the 29 anodes and the 28 cathodes were of lead. No trouble is mentioned with respect to the anodes, and it is presumed that they were peroxidized. The cathode area exposed was 38.5 (later 42.8) square feet for both sides and all electrodes were spaced one-third inch apart. Allowing one-eighth inch for thickness of lead, we have 1.66 cubic feet as the volume of box actually occupied by electrodes. The capacity which I claimed such a box should possess was 8.30 cubic feet per 24 hours.

Twelve experiments were made with a single flow through the box at a rate varying from 0.75 to 0.87 long ton, or 26.7 to 31 cubic feet per 24 hours, which is three to four times the capacity claimed for such a box. With 2.5 to 3 volts and 3.5 to 3.8 amperes and an average current density of about 0.1 ampere per square foot, the average of 12 experiments was a recovery of 32.3 per cent gold and 31.2 per cent silver, with a reduction in cyanide content of the solution from 0.142 to 0.138 per cent. If the volume treated had been reduced to 8.3 cubic feet per 24 hours, the results would, undoubtedly have been much the same as I have obtained.

SERIES B TESTS.

Three experiments were conducted with 6.1 cubic feet of solution passed four times through the cell in 3 hours, which would be at the rate of 48.8 cubic feet per 24 hours. With 2.5 to 3.5 volts and 3.5 to 10 amperes no gold was obtained. The current density here was 0.26 ampere per square foot, or nearly the same density as that used by Mr. Butters to deposit metal as a slime, and the rate of passage of the solution was about 5.88 times that which I have desired. The

velocity of flow of solution through the narrow slit between the plates, $\frac{1}{8}$ by 11 inches, would have to be more than 5 feet per minute to give this flow, hence it was to be expected that the loose slimy deposit would be stripped from the flat plates by the excessive flow.

SERIES C TESTS.

In this test the rate of flow was 0.92 long ton or 32.15 cubic feet per 24 hours, nearly four times the desirable capacity of the box. At 4 volts and 10 amperes and with a density of current amounting to 0.26 ampere per square foot the recovery of gold was 61.7 per cent and of silver 59.3 per cent, with no loss in cyanide. Considering that this is nearly four times the desirable capacity of the box, the results are not bad. The current density was sufficient to deposit slime on the flat electrodes, but the circulation was not sufficiently violent to wash it all off. Another experiment, at the rate of 0.9 long ton per 24 hours, gave 60.7 per cent recovery of gold and 59.8 per cent of silver.

In all the above experiments with flat electrodes the volume of solution was three to seven times greater than the capacity which I should have desired. As the duration of the tests is not mentioned the ampere-hour efficiency can not be calculated.

EXPERIMENTS WITH PERVIOUS WIRE-CLOTH CATHODES.

The solutions employed in the tests with pervious cathodes contained 7 to 25 pennyweights of gold per long ton, 1 to 3 ounces of silver, and 0.14 to 0.22 per cent potassium cyanide. The 28 cathodes and 29 anodes were spaced one-half inch. Each cathode was a single piece of 16-mesh iron-wire cloth, the total area being about 34 square feet. The anodes were of sheet lead resting on the bottom of the box and perforated by 80 one-half inch holes. These should have been five or ten times as numerous.

For the test a measured quantity of solution was placed in a barrel at the head of the deposition box, also filled with the solution. The current was allowed to run from the barrel through the box to a second barrel which received the tailing. At the end of the test the content of this tailing barrel was transferred to the head barrel. Later, after experiment 25, two head and two tailing barrels were used, to make sure that every part of the solution received its proper treatment.

Three interesting experiments, which will be here designated A, B, and C, were made to determine the effect of a stoppage of the dynamo. The conditions of test and results were as follows:

A. Solution standing in cell; dynamo stopped; brushes down; circuit not broken. Sample taken when dynamo stopped: Gold content, 5 pennyweight 5

grains; silver, 14 pennyweight 21 grains. Sample taken after standing 16 hours: Gold content, 6 pennyweight; silver, 16 pennyweight 5 grains.

B. Solution standing in cell; dynamo stopped; circuit broken. Sample taken when dynamo stopped: Gold, 15 pennyweight 12 grains; silver, 2 ounces 2 pennyweight 9 grains. Sample taken after standing 20 hours: Gold, 16 pennyweight 11 grains; silver, 2 ounces 5 pennyweight 20 grains.

C. Solution standing in cell; dynamo stopped; circuit broken. Sample when dynamo stopped: Gold, 5 pennyweight 23 grains; silver, 9 pennyweight 11 grains. After standing $1\frac{1}{2}$ hours: Gold, 5 pennyweight 21 grains; silver, 10 pennyweight 18 grains.

The longest stoppage that occurred during which the circuit was broken was one hour.

Allowing one-eighth inch for the thickness of the electrodes, which were one-half inch apart, the electrodes occupied 2,200 cubic inches, or 1.25 cubic feet. Thus the box was much less compact than those which I have used with wire-cloth electrodes, in which the current gap was never more than one-fourth and often less than one-eighth inch, and in which I never found indications of short circuits.

DISCUSSION OF RESULTS.

It will be observed that the best results of the experiments between Nos. 21 and 28 were obtained with rich solutions, test 24 giving the best results of all. The experiments ran too short a time, probably because the handling of the barrels was too onerous. The centrifugal pump would have removed that difficulty. If the cathodes were intermittently used, rusting and subsequent stripping must have taken place and lowered the results during the short time of the test.

In experiment 36 the wire cloth was 8-mesh. Only three sheets were placed between each cathode; hence adequate depositing surface was not provided. Though most of the solution passed beneath the cathodes, some must have passed through. The cathodes had evidently rusted with some gold and silver upon them from a previous experiment. This corresponds with my own experience. Then when the cathodes were used in experiment 36 the gold and silver immediately redissolved where the cathode had rusted, and until the rust was removed the current was wasted in reducing the iron oxide. The silver, being easier to reduce, came down first. Experiments 37 and 38 seem to show that the cathodes were in better condition; and though the current density is still high, the ampere-hour efficiency is fair. If these experiments had been conducted exactly as directed, the results obtained would have been better. Foul mill solutions are harder to treat than clean solutions. I have successfully treated many such solutions, as well as similar solutions saturated with lime. The solutions at the Waihi mine contained, among other substances, selenium, probably as selenate of sodium,

which is usually recovered with the gold and silver in the zinc clean-up. The real difficulty in the last experiments was doubtless the rusting of cathodes, through intermittent use. One of the cathode plates sent me, though carefully packed, was covered with rust.

EXPERIMENTS AT TRANSVAAL GOLD MINING ESTATES.

Some experiments were made with iron-wire cloth cathodes at Transvaal Gold Mining Estates, Pilgrim's Rest, South Africa, by A. Scrymgeour in August, 1905. The description given here is condensed from a report presented by him to W. W. Mein. The precipitation box used was of 1 cubic foot capacity, the cross section not being stated. The anodes were one-sixteenth inch iron plates with one-fourth inch holes. The cathodes were of wire gauze, the mesh, area, and construction not being stated. The first experiments were made with working solutions containing an average of 4 penny-weights gold per ton, copper as a double cyanide, and 0.016 per cent and 0.0087 per cent cyanogen and protective alkali. A long series of experiments gave an average precipitation of about 25 per cent of the gold after a half hour's contact.

The average precipitation of our ordinary Siemens and Halske box is 75 per cent for four hours' contact (eight times as long), so that the Christy method panned out appreciably better, but against that there is the fact of clean anodes and higher amperage. As there was, however, a considerable discrepancy between these results and Prof. Christy's, a clean solution of sodium aurocyanide was made up and a series of experiments done therewith.

The precipitation was most exceptional—75 to 80 per cent on half an hour's contact. With clean solutions of aurocyanide the iron plates were scarcely attacked, but with working solutions the plates became extremely foul with iron and copper cyanides and had to be cleaned up every week, the resulting "blue" containing 7.9 ounces to the ton, which is extremely high for so short a period.

With working solutions the free cyanide was consumed to the extent of about 25 per cent.

The very high rates of flow (1 foot a minute) mentioned in Prof. Christy's paper gave practically no precipitation. One foot in four minutes seemed most effective in our experimental box.

The wire-gauze cathode gave a very nice and clean deposit.

In this brief report some details are missing. The section of the cathodes, the skin area of their surface, the current density, the current, and the voltage, are not stated; therefore it is not possible to explain the low result obtained at the rate of flow of 1 foot per minute. With the foul solutions treated, iron anodes should not have been used, but, instead, perforated peroxidized lead anodes. The presence of copper, moreover, introduces difficulty in the precipitation of the gold by any process, as will be considered later.

EXPERIMENTS WITH PEROXIDIZED ANODES.

These experiments, like those previously described, were conducted in the latter part of the year 1905 by A. Scrymgeour on copper and gold bearing cyanide solutions. The lead plates were peroxidized in permanganate of potassium solutions and were one-fourth inch thick, with five-eighth inch holes. The holes were too large and were probably too few in number. The results were better than with iron anodes.

"The cathode gave, upon the whole, a good deposit, but a slight furring of the copper was noticed at the higher amperages." The solutions treated contained 0.046 to 0.065 per cent copper, 0.004 to 0.006 per cent alkali, and 0.0064 to 0.013 per cent cyanide. The rate of flow varied from one-eighth to one-third of a foot per minute, and the current was 0.014 to 0.033 ampere per square foot of cathode surface. The original content of gold was 1.05 to 2.2 pennyweights, which after one hour became reduced to 1.25 and 0.3 pennyweight, with 16 to 71 per cent recovery. In this and in other experiments there were evidences of cathodes having rusted from intermittent use.

In one test a clean solution of sodium aurocyanide was treated, the rate of flow being one-eighth foot per minute and the current 0.014 ampere per square foot. The original content was 2.2 pennyweights, becoming reduced after one-half hour to 0.40 pennyweight, and after one hour to 0.10 pennyweight, showing an extraction of 55 per cent. In only one or two tests was there any evolution of gas. The current density used by Siemens and Halske is given as only 0.006 ampere per square foot, and Scrymgeour, judging by that, evidently thought it best not to push the current density too far. The density, however, was not sufficient to insure a good precipitation. The best results were obtained with the higher current densities.

EXPERIMENTS AT ROODEPORT CENTRAL DEEP, 1905.**FIRST GROUP TESTS.**

In 1905 some experiments were made at Roodeport Central Deep mines, in South Africa, by Gabriel Andreoli. In these experiments, after some preliminary work with low current densities, similar to those usual in the Siemens & Halske process, the box was operated at such a density as "to just allow a slight evolution of gas to be visible at the electrodes." To quote further from Andreoli's report:

The results of this experiment being in my opinion very encouraging, it will perhaps be interesting if I give full details:

Space of box occupied by anodes and cathodes, slightly under 1.5 cubic feet, say 90 pounds.

Flow of solution through box maintained during the experiment at 4 pounds per minute (0.0642 cubic feet per minute or a velocity through box=0.77 inches per minute)=duty 64 times capacity of box.

Twelve anodes 1 foot square of $\frac{1}{8}$ -inch sheet iron perforated with $\frac{1}{4}$ -inch holes; 24 cathodes in pairs, each cathode consisting of five sheets of mill screening clamped and connected together, $\frac{1}{4}$ -inch space between anodes and cathodes. Current at start 10 amperes, slight evolution of gas apparent, current subsequently reduced in stages to 7 amperes, maintaining evolution of gas at electrodes.

KCy and alkalinity tested right through the experiment, and practically no alteration, KCy 0.068 per cent, alkalinity expressed as NaHO=0.0218 per cent.

Value of solutions:

Before first circulation, 8.04 pennyweights per ton.

After first circulation (drip sample at foot of box) 1 pennyweight per ton.

Recovery, 87.5 per cent. Duty, 64:1.

Before second circulation 3.66 pennyweights per ton.

After second circulation 0.6 pennyweight per ton.

Recovery, 81.4 per cent. Duty, 64:1.

Total recovery, 91.5 per cent. Duty, 32:1.

If it had not been for the accident of mixing the 1-pennyweight solution with some of the original solution I feel certain that the total recovery would have shown a more striking figure.

As it is I consider this very successful as compared with the Siemens & Halske process which in this plant gives an average recovery from solutions of only 70 per cent and the duty compared to capacity being 2.5:1.

The experiment, however, seems to contradict Prof. Christy's claims that the process is economical from an electrically efficient point of view. I may later on succeed in obtaining good precipitation with a lower current density, which was in this experiment extremely high, namely, 0.3 ampere per square foot of anode surface. Normal Siemens & Halske practice here is 0.04 ampere per square foot of anode surface.

If it is found that such a high current density is necessary for successful precipitation then insoluble anodes will be indispensable; but to any one accustomed to the so-called Prussian blue produced on a Siemens & Halske plant, its treatment is not a formidable matter.

In the three experiments the gold precipitated on the cathodes was in the form of a bright deposit with perhaps the exception of the last experiment, where the gold on the lower part of the box was inclined to be slimy, but this was to be anticipated, considering the high current density used.

In regard to the statement that the electrical efficiency is below that which I have found it to be in my own experiments, I may say that my own experiments are perhaps a sufficient answer and that no data were given to justify criticism. The voltage, the duration of the test, the quantity of solution treated, and the average number of amperes are not given by which the results might have been calculated. Moreover, it was afterwards found that the tanks used were old chlorination tanks already saturated with gold, which introduced a handicap that no process would overcome.

SECOND GROUP TESTS.

The following description of further experiments at Roodeport, in 1905, is from the report that Andreoli rendered to Mr. W. W. Mein:

From the results I have obtained so far it is noticeable that I have only had satisfactory results on the occasions when a heavy current density has been used, and the production of by-products due to the use of iron anodes has been considerable; and on the lines which I have been working the use of insoluble anodes appears imperative. I have communicated with you, and a set of peroxide of lead anodes is being made.

As you will notice from the results obtained in experiment 6, the duty of the box is considerably higher than anything the Siemens & Halske process has approached, besides being much more effective, but the competition to be met from the Siemens & Halske process is negligible; it is the ordinary zinc precipitation which has to be met. Here again experiment 6 gave an equal precipitation result at a higher proportion of daily flow to the capacity of the box than even the zinc process and with the expenditure of more electrical power. So far as I can see, the duty of the Christy box can be raised still further until a practical limit is reached. This is what Prof. Christy has claimed all along; large quantity of solution treated in a small cubic space, but I can not up to the present indorse the economy of electrical power or electrochemical efficiency. I propose at some future date to repeat a rapid circulation experiment, such as the patent specification insists on, but the result of my No. 1 experiment and Scrymgeour's experience are not encouraging.

In the last month's experiments the cathodes (made of mill screening) received under the high-current density conditions a remarkably hard and bright deposit.

The following is a summary of the experiments made this month:

Experiment 4.—Speed of flow increased to 6 pounds per minute. Value before first circulation, 3.1 pennyweights; after first circulation (dip sample), 0.4 pennyweight; recovery, 87 per cent; duty, 96:1.

Value before second circulation, 0.88 pennyweight; after second circulation (dip sample), 0.34 pennyweight; recovery, 61 per cent; duty, 96:1. Total recovery, 90 per cent; duty, 96:1.

Experiment 5.—Current high, as in Nos. 3 and 4; speed of flow increased to 10 pounds per minute. Duty, 160 times capacity of box. Evolution of gas only apparent at lower end of box, and eventually, as current dropped, it had to be increased to maintain evolution of gas at electrodes. Value before first circulation, 5.4 pennyweights; after first circulation (dip sample), 1.2 pennyweights; recovery, 77.77 per cent; duty, 160:1.

Value before second circulation, 1.52 pennyweights; after second circulation (dip sample), 0.54 pennyweight; recovery, 64.51 per cent; duty, 160:1. Total recovery, 90 per cent; duty, 80:1.

Experiment 6.—Considering that the experiments summarized above were very satisfactory from the point of view of duty, percentage of recovery, but that the lowest values of solutions were 0.34 pennyweight (experiment 4) and 0.54 pennyweight (experiment 5), this experiment (No. 6) was made on solutions of low value, to determine the possibility of reducing solutions to the low gold values which obtain in our current practice here.

Current strength as high as in Nos. 3, 4, and 5, namely, 10 amperes.

Rate of flow of solution, 4 pounds per minute. Value before first circulation, lost in assay office; after first circulation (dip sample), 0.46 pennyweight; before second circulation, 0.62 pennyweight; after second circulation (dip sample), 0.12 pennyweight; recovery, 80.64 per cent; duty, 64:1.

Experiment 7.—Since this experiment (No. 6), was fairly successful, experiment 7 was made at a higher rate of flow (6 pounds per minute); and a lower density, 5 amperes, which showed an evolution of gas right through the experiment, on the lower half of the set of electrodes. Value before first circulation, 0.5 pennyweight; after first circulation, 0.28 pennyweight; before second circulation, 0.52 pennyweight; after second circulation, 0.46 pennyweight.

This experiment I considered either salted, or incorrect, and the experiment was repeated.

Experiment 7a.—Evolution of gas profuse right through the experiment. Value before first circulation, 1.55 pennyweights; after first circulation, 0.44 pennyweight; before second circulation, 0.62 pennyweight; after second circulation, 0.78 pennyweight.

I have entered against this in my log book that the sample of second circulation must have been crossed; but now I believe that at the slow rate of flow of solution, the sample before circulation is incorrect, as the solution standing in the vats some 24 hours takes up more gold than the assay return shows.

Experiment 8.—Nos. 7 and 7a giving strong evolution of gas at comparatively low current, the rate of flow was increased to 6 pounds per minute. Results no good at all. Value before first circulation, 0.72 pennyweight; after first circulation, 0.54 pennyweight; value before second circulation, 0.56 pennyweight; after second circulation, 0.7 pennyweight.

Experiment 9.—Nos. 7, 7a, and 8 having given results of no value, No. 9 was made at the same rate of flow as No. 8, namely, 6 pounds per minute, and the current was increased to the strength used in No. 6, which gave a comparatively good result. Value before first circulation, 1.8 pennyweights; after first circulation (dip sample), 0.3 pennyweight; before second circulation (dip sample), (assay not done); after second circulation (dip sample), (assay not done).

I notice in one of your extracts from Prof. Christy's letters that he says he can reduce solutions to the value of 1 or 2 cents; but that he prefers to make it only 10 or 12 cents. This latter I have done in No. 6; but it is too high for our normal practice here, although the disadvantages might be outweighed (if it can not be overcome, which I do not believe) by the economy of initial outlay, efficiency, simplicity of clean-up, and other advantages inherent to the process.

DISCUSSION OF RESULTS.

In comment on the foregoing experiments, I can but express regret that old chlorination tanks had to be used. Evidences of the salting of the solutions at the most critical junctures are apparent. The omission of a record of the voltage and the amperage, hour by hour, of the duration of the experiment and of the volume of solution render it impossible to judge the ampere-hour efficiency. A two-barrel method of circulation is not satisfactory, for the reason that the velocity of flow diminishes with the diminishing head in the head tank, except when constantly regulated. The necessity of increasing the flow of solution as the liquid becomes impoverished of its gold and silver content can not be too positively emphasized.

It must be increased till the evolution of gas at the cathodes either ceases or becomes a minimum. Experimenters, by omitting these precautions, fail to obtain the results and the efficiency that are possible. It is true that gassing at the anodes need not be stopped, but some fail to appreciate the effect of the greater precipitating surface that I insist on. The best results are obtained by increasing simultaneously the cathode surface, the current density, and the speed of circulation. The current density should be kept at the highest point that will allow the metal to come down firmly coherent and to increase the circulation so as to reduce gassing at the cathodes to as low a point as is possible with the highest practicable current density. The limit of current density is reached when the metal comes down as a slime, and as it forms is washed off by the solution.

A difficulty that Andrioli must have met, though he does not mention it, was the intermittent use of the cathodes. From the dates of the experiments I observe that the interval was sufficient to cause the rusting of the iron wires beneath the gold. Some of the gold must have redissolved in the first part of his later experiments, and whether this was later corrected by the current can not be determined from the data given.

In Scrymgeour's experiments and Andrioli's first experiments the large amount of lime in the solution tested seems to have deterred them needlessly from employing adequate circulation. I have treated solutions rich in lime at velocities of flow of several feet per minute through gauze cathodes with no trouble from this source. With new cathodes trouble sometimes arises from a nonconducting coating of oil on the gauze or from the cathodes having been previously used and allowed to rust by intermittent use, as I believe was more probably the case.

I wish to express my hearty thanks to Messrs. Hopkins, Scrymgeour, and Andrioli for the trouble they have taken with these tests and my realization of the many limitations as to necessary conveniences and with constant interruptions. I consider the results favorable in view of the departure from the directions given and from the conditions that I would have desired.

SNODGRASS CYANIDATION APPARATUS.

A patent, of 10 claims, by J. Snodgrass (U. S. patent 835329, Nov. 6, 1906), appeared shortly after the experiments that have been described were made. The apparatus disclosed in the patent consists of a rectangular box through which the solution flows only once. The bottom of the box is covered with an iron plate, which is connected with the negative electrode. The cathodes are made of iron

frames resting on the bottom cathode plate, and either faced with iron-wire gauze of 100 meshes to the inch or with woven cloth of fine texture, coated with plumbago or with plumbago and a lead salt. The anodes are wooden boxes with coarse cloth sides, filled either with gas carbon or coke, graphite, binoxide of manganese, or peroxide of lead. The solution does not flow directly through the anodes, but is shunted around them. Carbons or other conductors are inserted into the box and connected with the positive pole. The iron plate connected with the cathode is claimed to have the advantage that any metallic powders falling on it from the cathode are prevented from being redissolved. The inventor expressly disclaims the use of rapid and repeated circulation of the solution through the deposition box, and urges this omission as an advantage.

It is apparent that effective precipitation under such conditions can be procured only by making the box large and costly and the clean-up difficult and expensive. To circulate the solution around and not through the anodes sacrifices one of the benefits of pervious electrodes. The iron plate at the bottom, on which the cathodes rest, is a good feature.

"COKE WHISKERS" AS AN ELECTRODE.

While I was engaged in a systematic search through all the known elements and compounds for a substance that would serve alternately as an anode and a cathode in cyanide solutions I received a specimen of the product popularly known as "coke whiskers." This substance was produced by Dr. J. A. Holmes in some of the experiments in coking coal that were undertaken by him at the Government testing plant at St. Louis, Mo.

The product closely resembles in size, shape, and color the hair of a negro's head. It is, however, a peculiarly resistant form of coke. It is but slightly attacked by the strongest nitric acid containing chlorate of potassium. It is a pervious mass, a good conductor of electricity, presents an enormous surface for deposition, and, in short, seemed to be the substance I had been looking for. It was not a newly discovered substance, but had been previously mentioned by Percy.^a He describes the substance as follows:

Hairlike form of coke.—Hairlike threads are sometimes observed on pieces of coke. They are nearly solid, and under the microscope occasionally present somewhat the appearance of a string of beads, soldered together. They consist of carbon, which seems to have been deposited in the following manner: A bubble of tarry or hydrocarbon vapor and gas, in escaping from the surface of the coke, becomes more highly heated, and is in consequence decomposed with

^a Percy, John, *Metallurgy*; fuel. 1875, p. 421.

the separation of solid carbon, which is deposited as a continuous coherent film on the surface of the bubble; a second bubble escapes through the carbonaceous shell of the first, and is similarly decomposed, forming a second carbonaceous shell attached to that resulting from the first bubble; and so, by a continuous succession of such deposits of carbon, a continuous tube of carbon is formed. Gas would continue to flow through this tube, depositing carbon in its course on the inner surface, until at length the tube is converted into a nearly solid fiber. Such appears to me the mode of formation of this curious hairlike matter, though I am by no means certain of the correctness of this view. I have noticed considerable variety in appearance in the specimens of hairlike coke, which I have examined under the microscope. This is a subject that deserves investigation by a competent microscopist.

From some accounts of the experiments at the testing plant at St. Louis I understand that few coals are capable of forming the product and that no one seems able to control the conditions that produce it. I tried to get quotations on the product in quantities. One coke manufacturer quoted it at \$20 per ton, but could not guarantee any quantity at that price. The subject, however, deserves further investigation, as there are many industrial uses to which such a material could be put.

EXCELSIOR CHARCOAL AS ELECTRODE.

The use of "coke whiskers" being out of the question on the large, commercial scale, I was forced to have resort to some other form of carbon that would be a substitute. It occurred to me that "excelsior fiber," a substance very generally used for packing and in the upholstering of cheap furniture, might be made into a pervious mass presenting a precipitating surface like zinc shavings.

Ordinary charcoal is practically a nonconductor, but I overcame the difficulty in this regard by heating the charcoal with metallic salts of silver, copper, or lead. I found that such charcoal could be converted into graphite in the electric furnace. Better still, I found that it was not necessary or even desirable to use the electric furnace, or to convert the excelsior entirely into graphite, as charcoal heated to a low orange became an excellent conductor without the addition of any foreign substance whatever. Grass, pine needles, straw, hemp rope, and many other materials also would serve, if coked at a sufficient temperature; but of all of these, excelsior charcoal proved the best. As regards ash content, for example, the excelsior was found to contain only about 1 to 3 per cent, whereas pine needles, for instance, often carry as much as 20 to 30 per cent.

I thus had a suitable material for pervious electrodes that could be used either as anodes or cathodes, or both in alternation. I found

that gold, silver, and copper could be precipitated upon such electrodes just as well as on a sheet of metal, and could be removed by making the charcoal cathode into an anode in a cyanide solution. The deposited metal could then be concentrated from a large cathode surface upon a small one. The most encouraging feature of all was that the charcoal of the anodes dissolved in the solution only at very high voltages, and after the gold and the silver had been removed. The action on the charcoal was much like that on graphite, and the charcoal, notwithstanding the large surface exposed, was less attacked than was hard carbon. Caustic lime seemed to precipitate any dissolved substance occasioning any discoloration; and after aeration, the solution so treated dissolved gold and silver as well as fresh solutions containing the same amount of cyanide would.

THE ELECTRODE SURFACE.

The area of surface exposed by "coke whiskers" and by excelsior charcoal is an important consideration. A cylindrical filament of "coke whiskers," 22 mm. long by 0.078 mm. in diameter and weighing 0.18 mg. has a skin surface, neglecting the area of the ends, of 5.388 square millimeters; therefore 1 gram of fiber represents an area of about 300 square centimeters. Thirteen grams of the material can be packed loosely, without difficulty, in a space of 384 cubic centimeters. Hence a gram occupies 30 cubic centimeters and has 300 square centimeters of precipitating surface.

"Excelsior charcoal" filaments may be produced of any degree of fineness. One average filament that I measured was 25 mm. long, 0.165 mm. thick, and 0.44 mm. broad. Its skin area would be 30.25 square millimeters, or 0.3025 square centimeters, and its weight was 1.6 mg. The surface of excelsior fiber thus may approximate 189 square centimeters per gram. The area for a given weight is then less than two-thirds that of an equal weight of "coke whiskers." Nevertheless "the excelsior charcoal," if made fine, is not so pervious to the solution, and on the whole works extremely well.

A comparison of the surface area of simple pervious electrodes as compared with the compound pervious electrode becomes of importance. When excelsior charcoal is placed in narrow rectangular boxes with cheesecloth sides to form electrodes that may stretch across the deposition box, two forms of construction are possible. If each box has a graphite or other conductor to supply the current we have what may be called a simple pervious electrode. If connected with the plus pole it becomes an anode, if connected with the minus

pole it becomes a cathode. Instead of making use of this arrangement, pervious anodes of another kind may be used. These may be graphite combs, perforated peroxide-of-lead sheets, or the peroxidized lead wire anodes previously described. Any combination of these makes a suitable working system. With the simple pervious cathode, having both sides exposed to an anode, the metal extends downward on both sides and penetrates the charcoal.

The simple pervious electrodes have given excellent results in the precipitation of metals from solutions, but have the disadvantage of requiring many electrical connections, one being required for each anode and for each cathode. As a consequence, whenever an anode or cathode is to be removed there is trouble in establishing and procuring the electrical connections. Constant watchfulness is then necessary to maintain good electrical contacts.

In figure 32 are shown two pervious cathodes, *c*; of the kind described. These may be supposed to be made either of wire cloth or of fragments of coke or graphite, or of "coke whiskers" or excelsior charcoal contained in a suitable box with sides of cheesecloth, and placed between pervious anodes, *a*. It would ordinarily be supposed that the whole of the inner surface of such

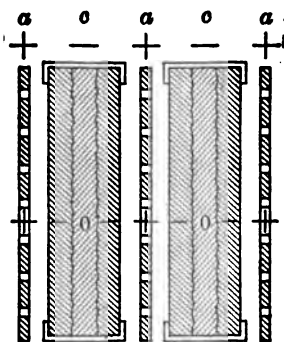


FIGURE 32. — Pervious electrodes. *a*, Anode; *c*, cathode.

pervious cathodes would be effective for receiving the precipitate of metal. However, I have found that metal precipitated by the electric current on such pervious cathodes forms a deposit of only moderate depth. The depth depends on the permeability of the material and increases with the free interstitial space. When granules of graphite, or of coke, one-eighth of an inch in diameter are placed in such frames, the metal deposit extends into the electrode to a depth of about 1 to 3 granules. With wire cloth it penetrates to a greater depth, but in all types, if the thickness of the electrode is great, the interior (marked 0, fig. 32) receives practically no deposit, while the outer portions, marked with the minus sign (—), become heavily coated. The interior may be said, then, to be in an "electrolytic shadow." Here and in what follows regarding this neutral portion of the pervious electrode, I refer only to the metal that is deposited by the electric current, and not to that deposited by chemical action of the solution on the material of the electrode itself, as sometimes happens. The depth through which the current penetrates and acts on the solution varies from one-eighth to

one-half inch with the ordinary perviousness. The more compact the particles of the conducting substance, the shallower will be the metallic deposit, until at the extreme, when the perviousness diminishes to that of a sheet of compact metal, the deposit does not penetrate the cathode at all and is deposited on the surface. The discovery of this fact, which from the first appeared to be a disadvantage in the use of pervious cathodes, led me to simplify the pervious electrodes for receiving the deposit of metal. This feature is illustrated in figure 33.

In figure 33 is shown a deposition box, *A*, which is to contain the solution of a metal salt, such as copper sulphate, or cyanide of gold or silver and potassium. At *a'* is shown an insoluble anode of carbon or platinum, and at *c'* a cathode also of carbon or platinum. Now,

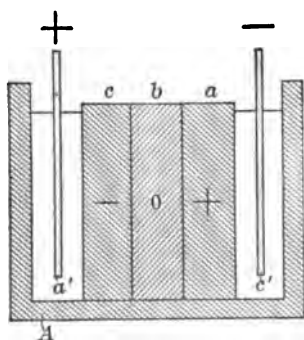


FIGURE 33.—Deposition box with pervious electrode. *a*, Anode (+); *b*, neutral part (0); *c*, cathode; *A*, deposition box.

if there be placed in the solution between these anodes and cathodes a pervious electrode, made either of sheets of wire cloth or a mass of granules of coke, graphite, "coke whiskers" or "excelsior charcoal," and if an electric current be caused to enter the solution at the anode *a'*, and to pass out by the cathode *c'*, the end of the pervious electrode *c* nearest the anode *a'* becomes charged with negative electricity, as indicated by the minus sign, and copper or gold and silver will be deposited upon that side of the pervious electrode to a depth of one-eighth to one-half inch or more, depending upon the perviousness. The oppo-

site side, indicated by the plus sign, of the pervious electrode becomes charged with positive electricity to a similar depth back from the face. The intermediate part, *b*, of the pervious electrode is neither negatively nor positively electrified; and practically no metal will be deposited there. My first improvement consisted in utilizing this idea in the construction of what may be called a compound pervious electrode. One face of this electrode acts as an anode and the other face as a cathode; between these faces is a part acting neither as a cathode nor as an anode, simply conducting the current as a solid metallic conductor would do.

Figure 34 shows the manner in which the compound pervious electrodes are used. Only the electrodes and not the deposition box are shown in this figure. The compound pervious electrodes are marked

e_1 and e_2 . They may be of any number desired. The positive electric current enters the solution by means of a simple pervious anode e , the whole of which becomes charged with positive electricity. At a suitable distance, preferably as small as possible without short-circuiting, is placed the compound pervious electrode marked e_1 . The side c nearest the anode e acts as a cathode and is marked with a minus (—) sign. The middle portion b is neutral and is marked o . The side a acts as an anode and is marked with a plus (+) sign. At a suitable distance is placed a second compound pervious electrode e_2 , and at proper distances as many more of these as may be necessary. Finally a simple pervious cathode e_3 , marked with a minus (—) sign, is provided where the electric current leaves the solution. The manner in which I prefer to arrange such a combination of compound pervious electrodes is shown in figure 35.

In figure 35, at B , is shown a large tank or reservoir containing the solution to be treated, and at A the deposition box. The solution is forced to circulate through the box A by means of the centrifugal pump F , driven by some such means as the motor G . The circulation is from the tank B through the deposition box, and back again repeatedly through pipe g . Also, solution may be drawn from the bottom of the tank and supplied to the tank on a float at the top, so that the poor solution above shall not mix with the rich solution below. It is also possible to circulate the solution from the tank through the box once only, provided the box is made sufficiently long to insure complete precipitation in a single passage; but the method shown in figure 35 is preferable. The electric current enters at the simple pervious anode e , passes through the solution to the compound pervious electrode e_1 , then through the solution again to compound pervious electrode e_2 , and so on to the simple pervious cathode e_3 , by which it leaves the box through the ammeter E . At C is shown a suitable voltmeter for determining the voltage between the terminals of the deposition box A . By the action of the current each of the compound pervious electrodes e_1 , e_2 , etc., receives a metallic deposit on the side that is negatively electrified. The side a , as shown in figure 34, receives no metallic deposit and the part b practically none, unless by the chemical action of the electrode itself, while the side c receives the deposit as a whole. The part marked a being subjected to such action as occurs upon any anode, is liable to be attacked by

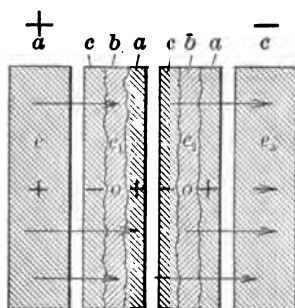


FIGURE 34.—Arrangement for compound electrodes. a , Anode (+); b , neutral part (o); c , cathode (—); e_1 , e_2 , compound pervious electrodes; e , simple pervious anode; e_3 , simple pervious cathode.

the solution unless made of some substance insoluble in the electrolyte.

In cyanide solutions "coke whiskers" and "excelsior charcoal" have been found sufficiently resistant to act as anodes at any ordinary voltage without protection. The aerated solution has the same solvent action upon gold and silver as does a pure solution of the same strength.

Such compound, pervious electrodes may also be termed pervious bipolar electrodes. They have a great advantage over the simple pervious cathodes. Thus a box requiring a potential drop of 2.5 volts with a 110-volt current can be arranged with 40 current gaps—

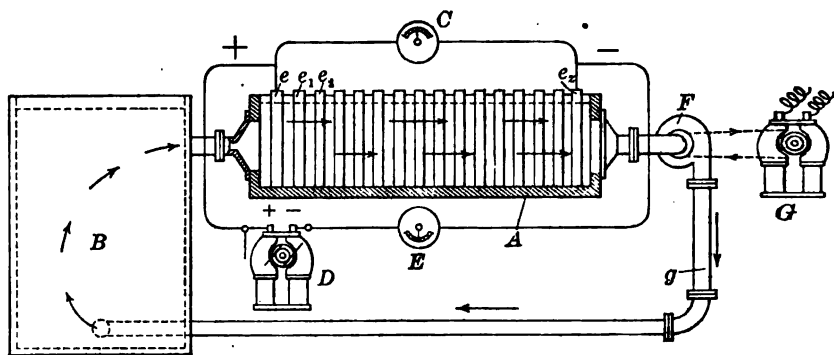
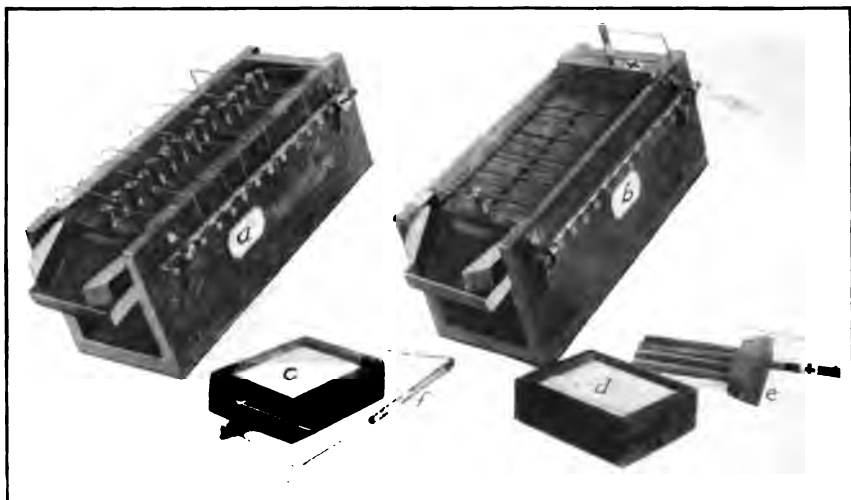


FIGURE 35.—Arrangement of apparatus for use with compound electrodes. *e*, Simple pervious anode; *e*₁ *e*₂, compound pervious cathode, and *e*₃, simple pervious cathode. Other letters same as in figure 30.

namely, 1 pervious anode, 39 compound pervious electrodes, and 1 simple pervious cathode. There being only two electric connections to be made, any intervening compound pervious electrode can be removed and replaced as often as convenient without interrupting the current. If the simple pervious cathode at the end is to be removed, an uncharged one can be placed in front of it and connected with the negative pole, when the other can be removed from the circuit. A direct-current service of 220 volts has been used, but I do not recommend so high a voltage, on the whole, on account of the inconvenience to those working at the deposition box. It is, however, easy to protect one's self from shocks by wearing rubber-soled shoes, or, if necessary, rubber gloves. The 110-volt current gives no inconvenience without protection. A patent (U. S. patent 883170, Mar. 31. 1908) was granted me for the electrodes herein described, and further details may be found in the claims there set forth.

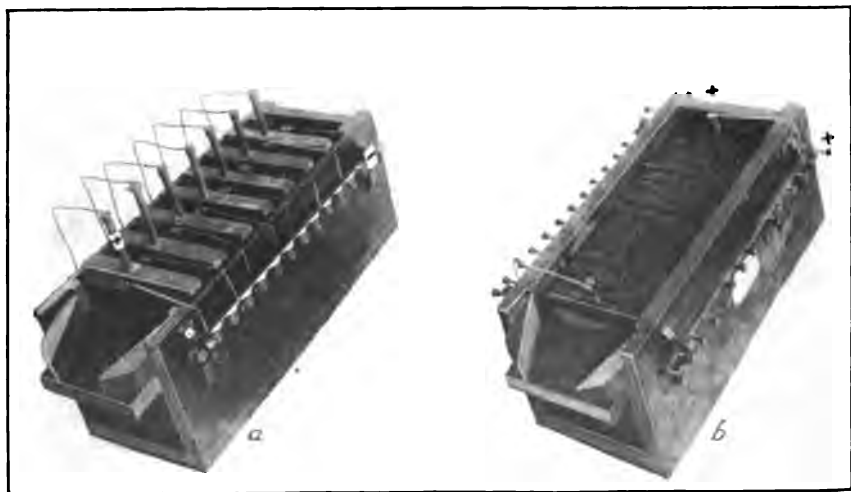


APPARATUS FOR ELECTRODEPOSITION OF GOLD AND SILVER FROM CYANIDE SOLUTIONS. CHRISTY PROCESS.



A. DEPOSITION BOX USED IN EXPERIMENTS.

a, Box equipped with 7 excelsior charcoal anodes and 8 charcoal cathodes; *b*, box equipped with 1 simple Acheson graphite anode (+), 13 compound charcoal cathodes, and 1 simple charcoal cathode; *c*, 2 simple electrodes; *d*, 2 compound electrodes; *e*, Acheson graphite anode; *f*, Acheson graphite rod $\frac{1}{8}$ inch thick.



B. DEPOSITION BOXES, ALTERNATIVE ARRANGEMENTS.

a, Box with 7 Acheson graphite comb anodes (+) and 8 simple excelsior charcoal cathodes (-); *b*, box with 1 simple charcoal anode (+), 13 compound charcoal cathodes, and 1 simple charcoal cathode.

DESCRIPTION OF AN EXPERIMENTAL PLANT.

As many readers may desire to verify the statements here made and to try the process on their own solutions, details are given here for complete small-scale tests. Such tests always should precede large-scale work, for by so doing the essential factors can be foreseen, and the necessary size of plant can be accurately determined. A type of plant used by the writer is shown in Plate I.

DEPOSITION BOX.

The early experiments were made with a box 10 inches in length, which afterwards was increased to 13 inches, as shown in the illustrations, and still later to several feet. The box is 6 inches deep and $4\frac{1}{2}$ inches wide and discharges at a height of $4\frac{1}{2}$ inches. It is illustrated in Plate II, A.

The box is made of redwood 1 inch thick. The end joints are put in with a tenon and two pieces of string are laid along the joints to bruise the wood, causing it to swell tight when wet. After being nailed, the box is immersed in water until it swells tight. It is then taken out and partly dried, after which it is given several coats of a special paint, such as "P & B" paint made from residues of California petroleum dissolved in carbon bisulphide. It is important not to apply the paint until the wood has swelled. If painted first, the box can never be made tight. Boxes fitted with circular-saw joints leak less, as a rule, than those with planed joints.

ELECTRODE BOXES.

The electrode boxes are made of strips of soft wood one-half inch thick and three-eighths inch wide. On the bottom is nailed a strip one-eighth inch thick and three-fourths inch wide. These are fastened together, and over the rear side is stretched a piece of cheesecloth, which is drawn tight, overlapping, and fastened to the box with thick "P & B" paint.

The outside dimensions of the frames as constructed are $4\frac{1}{2}$ inches wide by $5\frac{1}{2}$ inches high. The inside dimensions are $3\frac{1}{4}$ inches wide by $4\frac{1}{8}$ inches high. The box is half an inch deep. The inner cross section is practically 16 square inches or 1 square decimeter; that is, one-ninth square foot or 0.01 square meter. For making a new apparatus I should advise an inner cross section of 4 by 4 inches, making exactly one-ninth square foot, or 1 square decimeter, and should modify the dimensions of the deposition box accordingly.

The cover of these boxes is made three-eighths inch wide and one-fourth inch thick. Stretched over the inside and lapped over the outside is a layer of cheesecloth, which is fastened to the frame by a

coat of thick "P & B" paint. The cover is provided with a small pin at the bottom to be inserted in the projecting lip at the bottom of the frame, and is fastened on the top by a simple clamp made of hard rubber. The box, when thus made and assembled, holds approximately 8 to 10 grams of excelsior charcoal in a space 10 by 10 by 1.25 cm. in size. The box should be put together in such a manner that the cheesecloth side of the cover is in direct contact with the charcoal, thus leaving a space of one-fourth inch between the several boxes.

When these boxes are used as compound pervious electrodes they require no further addition, but when it is desired to use them for simple pervious cathodes an aperture of about one-fourth inch through the center of the box at the top is necessary for the introduction of the electrode rod of Acheson graphite. For simple pervious cathodes, however, a wire of iron, silver, or copper may be used, but the graphite rod is preferable. As thus constructed and assembled the box has a thickness of about three-fourths of an inch. Allowing for slight irregularities in construction, it is possible to use 15 of these boxes in a distance of 1 foot in length of the box; and as 1 of these serves as an anode, there are 14 effective compound pervious electrodes in each foot of length of box. Formerly I so made these boxes as to contain a layer of charcoal an inch thick, but found that better results could be obtained by reducing the thickness to one-half inch. Beyond this limit it does not seem to be wise to go. With the compound pervious electrode it is necessary to pack the charcoal as thick as indicated, namely, at least 10 grams to each box, to prevent passage through the charcoal without full chemical action. With two such boxes as I have indicated here—that is, with 28 current gaps—I have employed a voltage of 110.

CIRCULATING PUMP.

In order to give a small, compact precipitation plant, the use of a centrifugal or other circulating pump is necessary. The pump used in the small-scale experiments has been already described. For packing the stuffing boxes I use a little cotton or wool soaked in graphite and vaseline. The whole interior of the pump is painted with "P & B" paint, and where thus protected shows no signs of wear or chemical action. The only part of the pump which is not thus painted is the shaft, which is kept bright by the graphite and vaseline. I use two pumps on the same bedplate in preference to one in order to balance the side thrust.

The deposition box that I have here described has 4.28 liters capacity up to the water line. When filled with 15 electrode boxes with their charcoal content the clear space is about 2.4 liters. If the

compound pervious electrodes are not used, I prefer to use anodes made of Acheson graphite or the wire ones of peroxidized lead. In the small box are four Acheson-graphite rods about three-eighths of an inch in diameter. The upper ends of these are inserted in a graphite bus bar having conical or threaded holes an inch apart, thus giving an anode of comb-shaped form. I have used these graphite anodes repeatedly at 4 volts without signs of disintegration.

With simple pervious anodes I place an anode on either side of a cathode or between each pair of cathodes. With solutions containing sulphates I prefer to use the peroxidized-lead wire anode described on page 100. In this case I use a square frame made of painted wood three-fourths inch wide and one-fourth inch thick. Through this are stretched six lead wires about one-eighth inch in diameter, coated with peroxide of lead in the manner described. The wooden frame is coated with "P & B" paint.

In assembling the apparatus I connect the two suction pipes of the double pump at the opposite sides of the solution tank, preferably from near the bottom, rather than from the top, as in figure 35. The pump discharges through the deposition box, and the solution returns on the float board at the top of the tank, avoiding admixture with the stronger solution below. The circulation is maintained until the gold and silver content of the solution is sufficiently reduced.

I find the unit deposition box that I have described very convenient for use. The length of the box may be increased, if desired, or the same result may be attained by placing one box behind another, in series, until sufficient precipitating surface is obtained. With rich solutions containing much gold and silver, I have had the silver deposit so fast as to clog the filter cloth, a growth of silver having been formed on the outside of the cover, necessitating the removal of this from time to time to clear away the crystals of pure silver. This does not happen with poor solutions. However, as the interstitial space of the electrodes becomes filled with gold and silver, and particularly if the solution is foul with lime and ferrocyanide of potassium, often there is a considerable resistance to the circulation of the solution. There are two remedies for this, one is to reduce the speed of flow and the other to increase the grade of the box. I have frequently in precipitating rich solutions, on this account, inclined the box 1 to $1\frac{1}{2}$ inches to the foot, and have experimented with velocities of flow ranging from 2 to 5 liters per minute in the box of the size indicated. If the actual interstitial space in the box is about $2\frac{1}{2}$ liters, the actual rate of flow through the box would be 1 to 2 feet per minute. Such a rate, however, is not necessary with rich solutions. It will be noted in the foregoing that even greater velocities of flow may be required with dense currents and

low metallic content of solution to obtain rapid precipitation. When, toward the end, the solution becomes much reduced in metal the charcoal becomes choked with hydrogen bubbles, and the grade of the box must be increased or, with a high rate of circulation, the box will overflow.

DISTRIBUTION OF PRECIPITATION IN THE DIFFERENT CELLS.

When simple pervious anodes and cathodes are used, there is a uniform voltage throughout the whole deposition box, and the metal is precipitated on the outer side of each cathode uniformly throughout the box. With compound pervious electrodes a uniform distribution of voltage can be obtained if the charcoal is carefully packed so as not to leave any clear path for the electric current through the solution and not through the charcoal. There is always a tendency for the charcoal to settle in the box, and there is frequently a clear passage for the current along the water line or along the edges of the frame. In order to prevent this, care should be taken in packing the boxes. Moreover, electrode cells may be staggered by wedging them close to alternate sides of the deposition box, although it is preferable to avoid such grooves in a small experimental box, as they complicate cleaning in preliminary tests. In large boxes grooves are, of course, necessary. With proper precautions taken, the fall of voltage is very uniformly distributed throughout the box; but there is nearly always a tendency for the difference of potential between the first two boxes and the last two boxes to be greater than between the others. If the boxes are carelessly packed the difference may be considerable. On this account the first anode in the compound pervious electrode box should be a graphite comb or of peroxidized-lead wires. This distributes the current uniformly across the whole section of the box, and somewhat reduces the resistance.

If the distribution of the voltage throughout the box is uniform and the circulation of the solution through all parts of the box is also uniform, the precipitation of gold and silver will be very uniform. With the compound pervious electrodes the coating of silver seldom is more than one-eighth to three-sixteenths or at most one-fourth inch thick, but there is always a tendency for the simple pervious cathode at the end of the box to become more thoroughly coated than the others, owing to the leakage of current from the upper electrodes. For this reason it is good policy in treating very rich solutions to replace the last simple pervious electrode at the end of the box, as it becomes charged with silver and gold, more frequently than the others. The compound pervious electrodes also should be examined at intervals, and if the metallic precipitate tends

to creep up on the outside of the cover, giving danger of short-circuiting, it is best to remove the covers, replace them with fresh ones, and when convenient remove the silver from the coated covers. If desired the metal-laden charcoal may be replaced continuously, for no harm is done if any one of the compound pervious electrodes is removed from the series and replaced by another with fresh charcoal.

The experimental deposition box shown in Plate II, *A*, I have found most convenient. Boxes *a* and *b* are of the same dimensions and, as has been described, are 13 inches long on the inside and are capable of holding 12 inches of electrodes, allowing three-fourths inch for the distribution of the incoming solution at the head of the box and one-fourth inch for the discharge at the tail end. Box *a* is shown mounted with one form of simple pervious electrodes. There are seven simple pervious cathodes filled with excelsior charcoal and eight simple pervious excelsior-charcoal anodes. Each box is provided with an Acheson-graphite rod for the distribution of the current from the bus bar, the anodes being connected with a positive bus bar, and the cathodes with a negative one. At *c* are shown two of the simple pervious cathodes, one directly beneath the other. The only difference between the simple pervious anodes and the corresponding cathodes in this form of construction lies in the one being connected with the positive current and the other with the negative. At *d* are shown two of the same boxes used as compound pervious electrodes, the difference being that the electrodes *c* are provided with graphite rods for connecting them with the bus bar, whereas electrodes *d* require no such provision. However, as the simple pervious cathode has silver deposited on either side of the cloth, such cathodes are more convenient when made with a rectangular frame for the box one-half inch thick and with two covers one-fourth inch thick, provided with cheesecloth backing, so that the covers may be removed from either face of the electrode for removal of the charcoal. This construction is not absolutely necessary, but it makes the clean-up more convenient.

The box *b* shows the arrangement for a number of compound pervious electrodes and is the form that I prefer for that kind of work. At the head of the box, marked with a plus (+) sign, is shown the Acheson-graphite anode by means of which the positive current is introduced into the solution. This anode, shown at *e*, may be termed the Acheson-graphite comb anode. It consists of a bus bar of Acheson graphite into which are inserted four graphite rods, clamped together by means of graphite screws. The material is entirely of graphite except the little brass cap to which is fastened the copper connecting wire. The advantage of the Acheson-graphite anode is

that it more thoroughly distributes the current over the whole cross section of the box than would excelsior charcoal alone. As has been said, instead of the Acheson-graphite anode the peroxidized-lead wire anode may be used and is preferable if sulphates are present. In any event the intervening compound pervious electrodes *d* remain unchanged.

These boxes are as compact a form of construction as I have been able to devise of this type. In each box the solution is introduced into the box by the centrifugal pump, flows through the pervious electrodes, and out of the box again into the main tank. It will, of course, be understood that the grade of the box as shown in Plate II, *A*, is for convenience in representation. The grade that I prefer to use is about 1 to 1½ inches to the foot.

Alternative arrangements are shown in Plate II, *B*. At *a* is shown a very satisfactory form of simple pervious electrode, having seven simple, pervious, Acheson-graphite comb anodes and eight simple, pervious, excelsior-charcoal cathodes similar to *c*, Plate II, *A*, the cathodes being all connected with the negative bus bar and the anodes similarly connected to the positive bus bar.

The simple pervious electrodes shown at *a* in Plate II, *B*, are on the whole preferable to those at *a* in Plate II, *A*, as the anodes of graphite are less acted upon than those of charcoal. The graphite anodes may, of course, be replaced by those of peroxidized-lead wire.

An alternative form of compound pervious electrode is shown at *b* in Plate II, *B*. The current here enters the simple pervious anode marked with a plus (+) sign, passes through the 13 compound pervious excelsior-charcoal electrodes and out through the simple pervious cathode, marked with a minus (—) sign at the lower end of the box. As the simple pervious excelsior-charcoal anode is not so good a conductor as the Acheson-graphite or the peroxidized-lead-wire anode, it does not so evenly distribute the current across the head of the box, and, as stated, I prefer the Acheson-graphite comb anode or the peroxidized-lead wire.

Plate III, *A*, shows the apparatus equipped with the compound pervious electrodes such as was used to handle a 20-liter charge. The box, *b*, is of 4.28 liters capacity, and, with the electrodes, 2.4 liters. The box, together with the tank *d*, holds 22 liters of solution, a convenient charge for an experiment.

The positive current passes to the Acheson-graphite anode *a*, thence through the compound pervious electrodes, not shown, and out through the simple pervious cathode, *c*. The solution overflows from the box into the tank and then passes through the two pumps, *h*, and back again through the box, circulating until the extraction is completed. The motor, *g*, shown here for operating the pump is needlessly large. A compound pervious electrode box ready for use

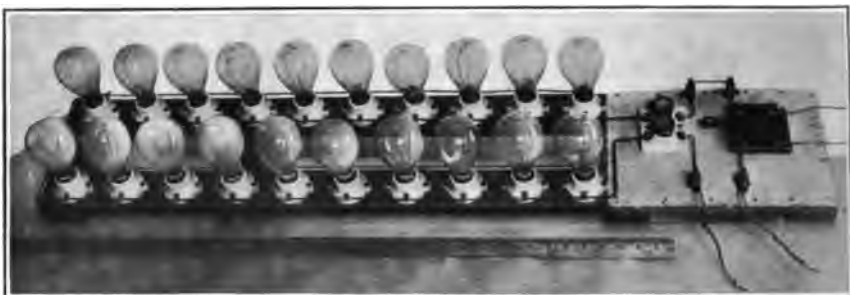


A. WORKING MODEL. CHRISTY PROCESS.

a, Acheson graphite anode; b, deposition box; c, simple charcoal cathode; d, solution tank; e, compound electrode holder; f, cover of holder; g, motor; h, pump; i, excelsior charcoal; jj, bottles of silver residue.

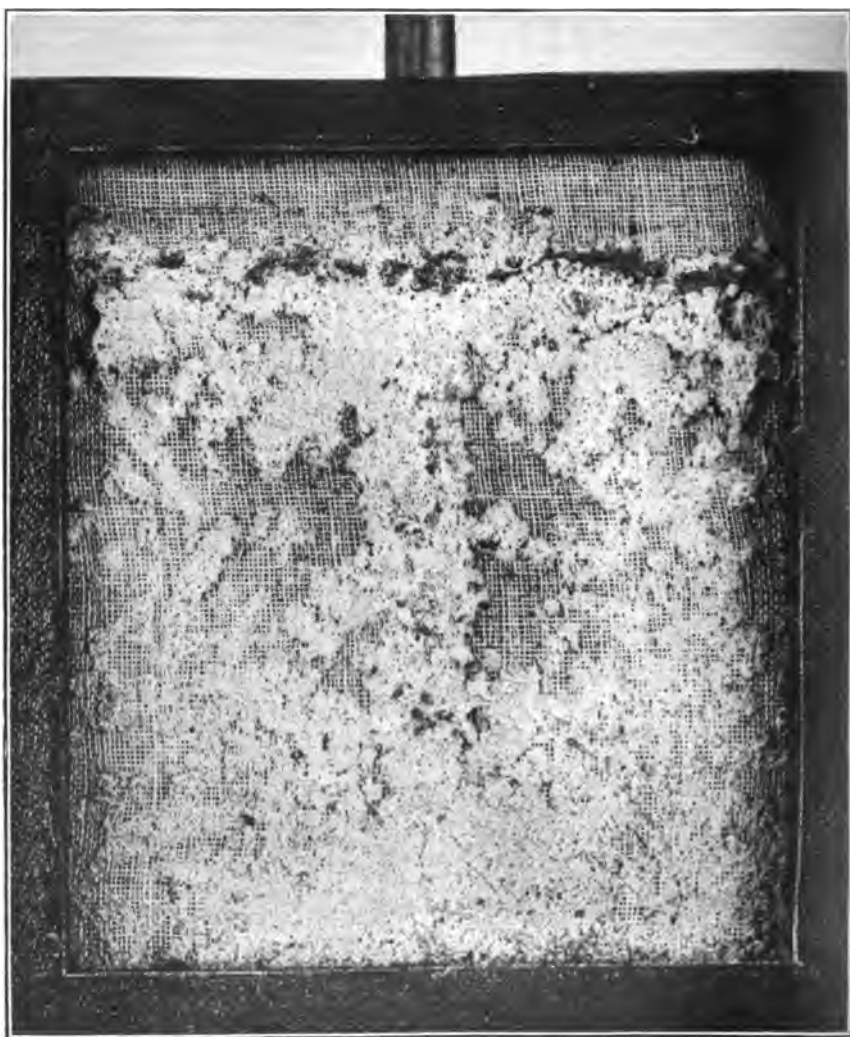


B. BANK OF LAMPS USED FOR RESISTANCE.



A. CONVENIENT RESISTANCE FOR EXPERIMENTAL WORK.

20 lamps in parallel, either 8, 16, or 20 candlepower as desired. Any number may be cut out as needed.



B. PERVIOUS CHARCOAL CATHODE, SHOWING CRYSTALS OF FINE SILVER ON CHEESECLOTH FRONT COVER.

is shown at *e*. At *f* is the cover of a compound pervious electrode, and behind this is seen the box itself filled with excelsior charcoal ready for use. The Acheson-graphite anode is shown at *a*. In the rear of the apparatus is a pan filled with excelsior charcoal as it comes from the retort. The two bottles, *jj*, in the foreground contain silver residues from the clean up and show the material as it appears after roasting.

Plate III, *B*, from a photograph taken in the old mining building at the University of California, shows the means of regulating the current. The laboratory is provided with a 110-volt direct current from a storage battery or from a dynamo. This current is brought into the laboratory at *a*, the main current passing through the resistances r_1 , r_2 , and r_3 . By connecting with varying points on this resistance, a shunt is formed with any desired voltage for the deposition box. The shunt current passes the ammeter *a* and the small adjustable resistance r and through the deposition box and back again to the resistance. A voltmeter to terminals of box is shown at *v*. The resistance r is used for adjusting the voltage accurately. By this means a slight waste of current results in the experiment, it is true, but it is possible to get any voltage needed from 110 volts to 1.

The system shown was resorted to after many experiments with small special dynamos, storage batteries, and other devices, and enables one to make use of a 110-volt direct current with no inconvenience. Moreover, by the means shown it is possible to regulate the voltage to 0.1 volt and keep it steady all day if necessary. It is necessary only to change the point of application of the shunt or throw in or out a few coils in r_1 or throw in or out a few of the lamps in the resistances r_2 , r_3 , making the final adjustments by the resistance r .

Plate IV, *A*, shows in detail the arrangement of a suitable shunt resistance. The frame carries 20 sockets, in which may be inserted in parallel electric-light bulbs of any desired candlepower supplying the resistance. These resistances are convenient and they are used generally about the laboratory.

MODE OF CHARGING THE BOXES WITH EXCELSIOR CHARCOAL.

The charcoal as it comes from the retorts is in the form of large ball-like masses of closely interlocked fragments. I loosen up the shreds of charcoal carefully so as to reduce the breakage to a minimum and form a mass of homogeneous threads. The material is then carefully charged into the boxes. In order that the charging may be uniform I prefer to counterpoise the box on a weighing scale and to add the charcoal until the required weight is obtained. With

compound pervious electrodes I prefer about 5 pounds of charcoal to the cubic foot of cathode space, and for simple pervious electrodes about $2\frac{1}{2}$ to 8 pounds. As I have already stated, the fine charcoal is put at the back or the anode side of the compound pervious electrodes, care being used to fill the box so that no clear space is left for a direct passage of the current. This is particularly necessary with the compound pervious electrodes, the aim being to pack it so that on being held up to the light no light is seen. When so packed the "electrical shadow" or neutral part will also be complete. This leaves 80 to 90 per cent of the interior of the box for solution and deposited metal. The material should be packed a little harder along the edges than in the middle. An inner diaphragm, mentioned in the S. B. Christy patent specifications, I have found unnecessary, provided the charcoal is carefully packed along the edges. A little "P & B" paint at the edges of the cheesecloth prevents electric leakage at these places.

PERMANENCE OF EXCELSIOR CHARCOAL.

If the excelsior charcoal is burned at an orange heat for an hour after the escape of gas ceases it has a steel-blue color, and I find that in this condition it is very little acted upon by the current even when used as an anode. So long as the silver and the gold are contained in the solution the charcoal is acted upon only in the very slightest degree; but after the gold and the silver have been precipitated, particularly if much caustic potash is present in the solution, the charcoal is attacked at a high voltage and slowly imparts to the solution a slight color like that of very weak tea. This coloring matter can be entirely removed from the solution by quicklime, but such removal is not necessary. I have found the action of the solution to be unimpaired if the solution is thoroughly aerated before being used as a solvent for gold or silver.

DETAILS AND RESULTS OF TESTS.

To leave no doubt as to the possibilities of the process I shall go somewhat into detail in certain experiments made by me on various gold and silver bearing solutions. The form of tables which I have adopted I believe to be the briefest and clearest. The conditions as to the circulation of the solution are not always specified, but it should be understood that the rate has usually been 1 foot per minute. In some tests a lower velocity has been found to suffice with solutions very rich in gold and silver, but with an impoverished solution it is well to increase the circulation to the rate specified to attain a satisfactory rate of precipitation.

The capacity of the boxes containing the charcoal electrodes is not quite so great as of those containing the wire-gauze electrodes. On account of the permanence of the charcoal, however, in first experiments it is better to treat charges of 20 liters at a time until familiar with the manipulations. Results can be obtained then in a single day's work and samples taken in the uninterrupted course of the precipitation. Before large-scale work is begun I should recommend charges of about 500 liters per day, and continuous operation. In this event samples need not be taken so often; perhaps once in four to six hours will suffice. The rise in cyanide content when rich silver ores are being treated will not be as rapid as in the small-scale work, for the reason that the large volume of solution treated requires a longer time for extraction. The rate of increase ought to be inversely proportional to the volume of solution.

In treating ores containing much silver it is not necessary to assay continuously, for if a sample of 1 c. c. of the solution is taken, and to it is added 1 c. c. of dilute sulphuric acid a precipitate of cyanide of silver shows silver to be still present in the solution to the amount of 4 to 5 mg. per 100 c. c. From the density of this precipitate the silver content can be closely estimated by eye. The rate of precipitation, in treating rich silver ores, can also be determined, if the voltage is uniform, by the fall of amperage. So long as the silver is present the current will be high, but as soon as the silver content of the solution falls the current also falls. As regards silver, results plotted on cross-section paper clearly indicate, by the form of the ampere curve, the point where the silver is substantially removed. With gold solutions this does not apply so strictly. The most economical point to stop precipitation is after the cyanide content ceases to increase and begins to decrease. I strongly recommend the plotting on cross-section paper of results obtained in all experiments. Such representation shows the facts in a striking manner and gives a better idea than can be obtained by examining numerals.

TAKING SAMPLES.

For ordinarily rich solutions 100 c. c. portions of solution have been used as samples. The sample is evaporated in a lead boat at a gentle heat with a good but not a strong draft. The evaporation takes place in an hour. The ears of the boat must not be folded tight or capillarity will cause the solution to climb and dry on the sides of the boat, leading to mechanical losses and low results. The dried residue in the boat is sprinkled with a little boracic acid, scorified, cupelled, and parted in the ordinary way. For very low grade gold solutions I prefer to take 603 c. c. of solution for evaporation, and, after scorifying, cupelling, and parting, to weigh the button of

gold in milligrams. Then every milligram represents \$1 in gold per ton of solution, and the amount is obtained directly in dollars and cents. This weight of 603, or more accurately 602.9 grams or approximately 602.9 c. c. of solution, I call the "California gold ton." With low-grade solutions a liter sample may be desired.

I always prefer the evaporation of the solution to any other method in use, as it is absolutely reliable if care is taken to avoid mechanical loss. With 100 c. c. samples results are obtained in about 2 hours. The evaporation method is to be preferred, but the following method of assaying cyanide solutions is satisfactory also, except for solutions very rich in cyanide:

To 100 c. c. of cyanide solution add 6 to 7 c. c. of a 10 per cent lead acetate solution, then add 3 grams of zinc shavings, heat just short of boiling, add 200 c. c. of strong hydrochloric acid, and heat again just below boiling so as not to break up the lead sponge which forms. When the zinc has been nearly dissolved, remove the precipitate of spongy lead, squeeze as dry as possible upon a piece of test lead, dry gently upon the hot plate, wrap with the test lead into the form of a ball and place upon the cupel. The time required to prepare the sample for the cupellation is 15 to 20 minutes. This method, suggested to me by Mr. F. J. Buel, I have found to be particularly satisfactory for solutions poor in gold and silver and not too high in cyanide. When there is a large amount of free cyanide present the results are sometimes low. Where the gold and silver content of the solution is very low and it is difficult to recover the gold bead, which is sometimes very small, it is sometimes an advantage to add 1 c. c. or more of a standard silver nitrate solution before precipitation. The silver nitrate will be precipitated on the lead and add to the size of the bead, preventing loss on cupellation. If desired to estimate the silver, one can deduct the value of the silver nitrate added.

USE OF AMMETERS AND VOLTMETERS.

It is absolutely impossible to do any satisfactory work with electrical precipitation without having constantly in the circuit a reliable ammeter and a voltmeter at the terminals of the cell. It is absolutely necessary to know what amount of current is passing through the solution. When these instruments are used, short circuits are instantly detected by the fall of potential and the rise in amperes, and it is desirable to record these results, as I have done, to know what is happening.

ADVANTAGE OF USING METRIC VALUES.

In investigations of this sort it is important to express results in simple form and to arrange the simultaneous variables in commen-

surate units. The metallurgy of the precious metals has been greatly retarded by the antiquated method of reporting gold and silver in ounces, pennyweights, and grains Troy, with other variables in pounds avoirdupois or in that uncertain unit, the ton, which may mean either the short ton of 2,000, the long ton of 2,240, or the metric ton of 2,204.6 pounds. The American custom of reporting gold in dollars and cents is not much better, for there is always the uncertainty as to whether the silver has been figured at variable current prices, or at one dollar an ounce. The natural difficulties of the problem are numerous enough. No one who has endeavored to correlate work done in Korea, Australia, South Africa, Mexico, South America, and the various American regions can fail to desire the elimination of all complexity possible.

The use of the metric system, estimating large values in metric tons of 1,000 kilograms, and small values in kilograms, grains, or milligrams, has an enormous advantage. Values for volumes and specific gravities are easily converted into definite weights without the uncertainties of the various incommensurate English units. For instance, if gold and silver are reported in milligrams per 100 c. c. of solution, ten times that number gives the number of milligrams per liter (practically per kilogram) and the number is at once the number of grams of gold and silver per metric ton of solution.

If the milligram weight of gold per 100 c. c. is multiplied by \$6.03 we have approximately the assay value of the gold solution per ton in dollars and cents. If the milligram weight per 100 c. c. of either gold or silver be multiplied by 0.29166+, the number of ounces per ton of 2,000 pounds will result. Or, if we multiply by 0.3 we have the approximate number of ounces of metal per ton of solution.

Of these methods of expressing values, the most useful is the expression of grams of gold and silver per metric ton of solution. This method, already widely used in Mexico and in most Spanish-American countries, is much more rational than the one used in this country.

IS COMMERCIAL CHARCOAL SATISFACTORY FOR PERVIOUS ELECTRODES?

In order to settle this point, commercial oak charcoal, used in the laboratory as fuel, was tested with 50 volts. No spark resulted and no current passed as measured by a milliammeter. One piece only that had evidently been overheated showed a trace of a spark at 50 volts. This piece was rejected and the remaining 74 grams was crushed to between 6 and 8 mesh size and filled into a box having cheesecloth sides. This was made a cathode by inserting a graphite rod connected with the negative pole. One graphite comb anode was used. Three liters of a cyanide of silver solution with 0.1 per cent

KCy and 0.1 per cent KHO were then circulated through the box for six hours, with the following results:

Results of test with oak-charcoal cathode.

Time, hours.	Volts.	Ampere.	Silver, mg. per 100 c. c.
0.....	3.5	0.060	126.64
3.....	3.5	.055	119.72
6.....	3.5	.082	106.02

This experiment proved beyond a doubt that such charcoal was practically a nonconductor. The silver did not precipitate on the charcoal, to any perceptible extent, but came down firmly on the graphite conducting rod, and collected slightly between the grains of charcoal. There may have been some chemical action by the charcoal, but none was visible, it came out as black as when it went in. The charcoal not only did not help but actually hindered and acted as would so much nonconducting sand adding to the electric resistance.

RESULTS OF TEST WITH EXCELSIOR CHARCOAL CATHODES.

Three liters of 0.1 per cent KCy solution containing 0.1 per cent KHO, was treated with a single pervious cathode of 10 grams of excelsior charcoal, which had been ignited at a low yellow heat. The solution was not as strong as had been used before, but all other conditions were similar. The results were as follows:

Results of test with excelsior-charcoal cathodes.

Time, hours.	Volts.	Ampere.	KCy content, per cent.	Silver, mg. per 100 c. c.
0.....	3.5	0.275	0.097	74.36
1.....	3.5	.320	.1055	
2.....	3.5	.360	.1195	
3.....	3.5	.285	.1430	.32
4.....	3.5	.258	.138	.10
6.....	3.5	.249	.111	.05

The precipitation was practically complete in four hours, which is in decided contrast with the results shown of the previous experiment. The high current with the low voltage was due to the high electrical conductivity of the excelsior charcoal.

TO WHAT EXTENT DOES EXCELSIOR CHARCOAL ALONE PRECIPITATE SILVER?

In this experiment 3.3 liters of 0.1 per cent KCy solution, containing 0.1 per cent KHO, were circulated through 14 grams of excelsior

charcoal with no electric current passing. The results were as follows:

Results of test with excelsior-charcoal cathode and no electric current.

Time, hours.	Volt.	Ampere.	KCy content, per cent.	Silver, mg. per 100 c.c.
0	0	0	0.0995	124.40
2	0	0	.0985	122.64
4	0	0	.0945	120.78
5½	0	0	.0935	119.84
9½	0	0	.0935	110.86

The 14 grams of excelsior charcoal had precipitated about 1 per cent of its own weight of silver. The loss of cyanide was small and no regeneration occurred as in the experiment in which electric current was employed.

LOSS OF CYANIDE ON CIRCULATING THE SOLUTION.

Following the previous investigation, experiments were undertaken to ascertain whether a loss of cyanide occurred from circulating the solution with the centrifugal pump. It was found that no such loss took place in the circulating solution when no electric current was passing, provided protective alkali was present, but that with no alkali a slight loss of cyanide occurred in eight hours, evidently due to the presence of carbonic acid from the air.

The cyanide titration each hour gave the following results:

Results of test for cyanide loss from circulation.

	Cyanide, per cent.
Beginning of test.....	0.1
End of first hour.....	.0990
End of second hour.....	.0985
End of third hour.....	.0975
End of fourth hour.....	.0965
End of fifth hour.....	.0955
End of sixth hour.....	.0945
End of seventh hour.....	.0935
End of eighth hour.....	.0930

EXPERIMENT ILLUSTRATING USE OF SIMPLE PERVIOUS CATHODES.

An experiment was undertaken to show the recovery of silver and of cyanide of potassium from a rich solution. Only four simple pervious cathodes were used. The current gap was half an inch. The total length of electrodes in the box was 9½ inches, and the

active cross section of the cathodes was 1 square decimeter, or one-ninth square foot, which is the active cross section of the electrodes in the following experiments.

The precipitation in four hours was 92.24 per cent of the silver, whereas the cyanide content increased by 0.090 per cent. A deposition box constructed on the same scale and holding 1 ton of solution, at the same rate, would have treated 36 tons in 24 hours, and would have precipitated 92.24 per cent of the silver.

RECOVERY OF CYANIDE.

The experiment was conducted with four simple pervious cathodes each 1 inch thick, and with a half-inch current gap. The cathodes were of excelsior charcoal and had a total nominal area of 8 square decimeters, or eight-ninths square foot, but the actual surface area, which was incalculable, was much greater. The rate of flow was 2.4 liters per minute. The current density was less than 2 amperes per square foot. The solution was one of KAgCy_2 , and contained 24 grams of silver. The results are shown in the following table:

Results showing the progressive regeneration of cyanide.

Hours.	Volts.	Amperes.	Silver content.		KCy per cent.
			Mg. per 100 c. c.	Ounces per ton.	
0	3.5	2.0	120.15	35.06	0.099
1	3.5	1.9	95.68	27.89	.113
2	3.5	2.15	68.26	19.90	.140
3	3.5	2.32	37.19	10.85	.168
4	3.5	2.25	9.48	2.77	.188
5	3.5	2.2	1.15	.34	.182
6	3.5	2.05	.08	.02	.170
7	3.5	2.00	.02	.006	.164

The preceding table shows that the most economical point at which to stop the precipitation was reached in four hours, for at that time, with only four cathodes, 92.24 per cent of the silver was precipitated and the cyanide had increased from 0.099 per cent to 0.188 per cent, or about 90 per cent. The amount treated was 20 liters, or 6 times the volume of the box, giving a capacity of 26 tons per 24 hours for a 1-ton box. A further analysis of the results of the above experiment is shown in the following table:

Analysis of results of experiment with simple previous cathodes.

Time, hours.	Volume of solution treated, liters.	Silver deposited.		Average ampere-hours.	Theoretical weight of silver deposited, grams.	Ampere hour efficiency, per cent.	KCy, per cent.	KCy, gain or loss.	
		Mg. per liter.	Grams.					Hourly, grams.	Total gain or loss, grams.
1	19.79	244.7	4.843	1.95	7.868	61.53	+0.019	+3.79	
2	19.68	274.2	5.369	2.005	8.070	66.70	+ .022	+4.33	
3	19.67	310.7	6.081	2.235	9.018	66.73	+ .028	+5.48	
4	19.46	277.1	5.391	2.285	9.220	57.58	+ .020	+4.28	
5	19.35	83.3	1.612	2.225	8.978	17.95	- .006	-1.16	+17.88
6	19.24	10.7	.206	2.125	8.574	2.40	- .012	-2.31	
7	19.13	.6	.011	2.025	8.171	.13	- .016	-3.06	
									- 6.53

The preceding table shows that the ampere-hour efficiency is high for the first four hours, and that if the action had been stopped 92.24 per cent of the silver would have been recovered. The efficiency of silver deposition was still fair in the fifth hour. For the entire seven hours the silver precipitation was 99.98 per cent. The last two or three hours are seen to have been wasteful of the electric current. The increase in current at constant voltage is probably brought about

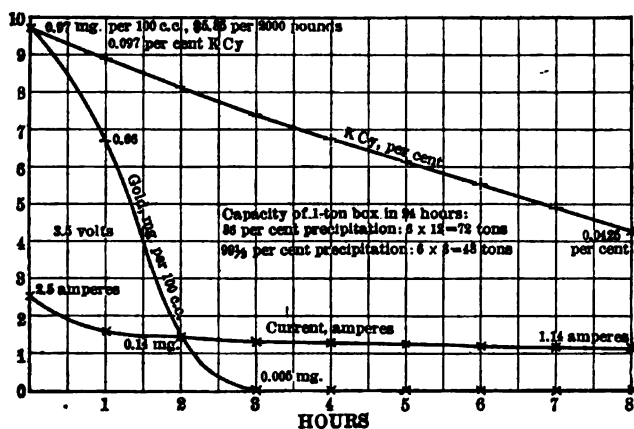


FIGURE 36.—Electrodeposition results with moderately rich gold solution; 4 simple previous cathodes, each 1 inch thick and containing 14 grams "excelsior" charcoal, and 5 Acheson-graphite comb anodes, each with 4 4-inch rods. Length of box occupied, 9½ inches, cross section 1 square decimeter, 8 cathode faces. Voltage, 3½, ½-inch current gap. Solution, 20 liters, contained 20 grams KCy, 20 grams KHO, 194 mg. Au as KAuCy_2 . Rate of flow, 8 liters per minute. Compare with fig. 37.

by increased conductivity of the charcoal due to the deposited metal on it, and the final fall marks the point when the silver was nearly all deposited from solution.

TREATMENT OF A GOLD CYANIDE SOLUTION OF MODERATE RICHNESS.

An experiment was conducted with gold cyanide solution instead of silver cyanide solution, but all other conditions were identical

with the preceding one. The original content of the solution was \$5.85 in gold a ton. The results are shown in figure 36 and in the following table:

Results of experiment with gold solution.

Hours.	Volts.	Amperes.	Gold, milli-grams per 100 c. c.	Gold per ton.	KCy, per cent.
0	3.5	2.5	0.97	\$5.85	0.097
1	3.5	1.6	.66	4.02	.089
2	3.5	1.47	.14	.84	.081
3	3.5	1.38	.005	.03	.0735
4	3.5	1.29	Trace.	Trace.	.0675
5	3.5	1.25	Trace.	Trace.	.061
6	3.5	1.19	Trace.	Trace.	.0545
7	3.5	1.18	Trace.	Trace.	.0485
8	3.5	1.14	Trace.	Trace.	.0425

Evidently there was no advantage in continuing after the first three hours, as 3-cent tailings are sufficiently low. While a regeneration of cyanide is possible from rich silver solutions, it is not usually possible with such gold solutions as occur in practice. A steady loss

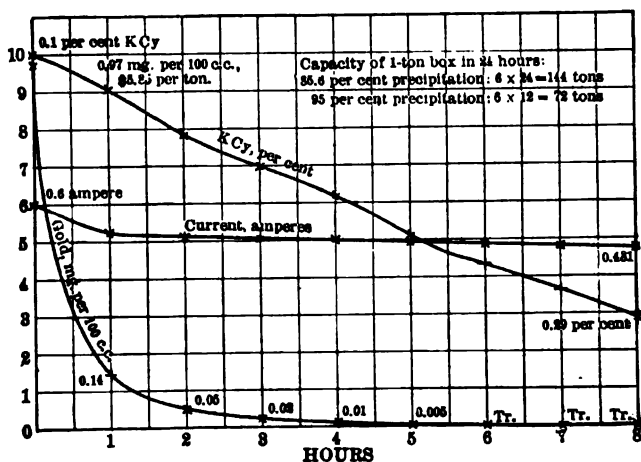


FIGURE 37.—Electrodeposition results with moderately rich gold solution; six compound pervious electrodes, each 1 inch thick and containing 14 grams "excelsior" charcoal, and 1 Acheson-graphite anode. Length of electrodes, 9½ inches; six cathode faces; six ½-inch current gap at 3.5 volts, 21 volts in all. Solution and rate of flow, same as in fig. 36.

of cyanide occurs with these, nearly proportional to the time, but increasing somewhat with the decrease of gold.

EXPERIMENT WITH USE OF COMPOUND PERVIOUS ELECTRODES.

A duplicate of the preceding experiment in all conditions other than that compound pervious electrodes were employed was next conducted. (See fig. 37; compare with fig. 36.) The five compound

pervious cathodes were each 1 inch thick. The length of the electrodes in the box was $9\frac{1}{2}$ inches. In this experiment there were only six cathode faces instead of eight, as before. The current gap was one-half inch. The six current gaps at $3\frac{1}{2}$ volts amounted to 21 volts. In three hours 98 per cent and in four hours 99 per cent of the gold was precipitated at a capacity of 24 tons per 24 hours in a 1-ton box.

RECOVERY OF SILVER AND REGENERATION OF CYANIDE.

A similar experiment was further undertaken to illustrate the recovery of silver and also the recovery of cyanide. In the experiment six compound pervious electrodes were used, each 1 inch thick and containing 14 grams of charcoal, and one simple pervious Acheson-graphite anode. The six current gaps were one-half inch each. The length of box was $9\frac{1}{2}$ inches, and the solution was 20 liters in volume, containing 20 grams KCy, 20 grams KHO, and silver in the form of KAgCy_2 . The results were as follows:

Results of experiment to show recovery of silver and cyanide regeneration.

Hours.	Volts.	Ampere.	Silver.		KCy, Per cent.
			Milligrams per 100 c. c.	Ounces per ton.	
0	21	0.730	125.29	36.6	0.0985
1	21	.709	86.97	25.37	.1240
2	21	.691	44.86	13.01	.1570
3	21	.622	16.21	4.72	.1795
4	21	.560	1.68	.48	.1830
Added KAgCy_2	21	.820	126.95	37.00	
5	21	.820	79.60	23.21	.219
6	21	.813	30.36	8.85	.252
7	21	.759	5.38	1.57	.251
8	21	.740	.38	.11	.236

The table shows that during the first four hours there was recovered 98.69 per cent of the silver from the 36.6 ounces present and 0.0845 per cent of the KCy, not allowing for samples. During the second four hours 99.7 per cent of the silver was recovered from the 37 ounces present, and 0.053 per cent of the KCy.

EFFECT OF CURRENT GAP.

Some experiments were undertaken to determine the effect of current gap upon capacity. In these tests the electrodes were one-half inch thick and were filled with excelsior charcoal. In the experiment in which, for example, the current gap was one-half inch, each box was one-half inch thick, being filled with charcoal filaments, and had on the outside a one-fourth inch cover, in such a way that two of these one-fourth inch covers were face to face, making a

current gap of one-half inch. The maximum capacity for a 95.5 per cent precipitation with one-half inch current gaps was found to be only 48 tons for a 1-ton box in 24 hours. With the one-fourth inch current gap the maximum capacity for 95.8 per cent precipitation was 144 tons, or nearly three times as great. The length of electrodes in these experiments was $9\frac{1}{2}$ inches. There were in the first experiment eight current gaps of one-half inch at $3\frac{1}{2}$ volts, making a total of 28 volts. In the second experiment there were 11 current gaps of one-fourth inch each at $3\frac{1}{2}$ volts, making a total of 38 volts.

The results of the two experiments, for which curves of precipitation are shown in figures 38 and 39, are well worth careful study and comparison. I strongly advise that the attempt be made to use the smaller current gap wherever it is possible to do so.

EFFECT OF LIME ON SILVER PRECIPITATION.

An experiment was conducted to determine the effects of lime on the precipitation, the solution containing cyanide of silver and free KCy. Compound pervious electrodes were used and one simple pervious Acheson-graphite comb anode. There were 11 cells, each one-half inch thick, and 11 one-quarter-inch current gaps of $3\frac{1}{2}$ volts each, giving $38\frac{1}{2}$ volts for the 11 boxes. The 20 liters of solution contained 20 grams of KCy, and 10.2 grams of CaO, together with 24.43 grams of silver as KAgCy₂. No caustic soda or potassa was present. The precipitation was 93.5 per cent complete at the rate of 144 tons per 1-ton box for 24 hours, and 99.5 per cent complete at the rate of 72 tons per 1-ton box in 24 hours. The results are shown in figure 40.

It is evident from the fall in the content of protective alkali that the caustic lime was partly converted into cyanide of calcium. It was noticed, further, that the fall in protective alkali was coincident with the rise in free cyanide, and that when the silver cyanide ceased to be precipitated the protective alkali remained nearly constant. When a new charge of cyanide of silver was added the protective alkali again fell and the free cyanide increased. The regeneration of free cyanide, in short, was at the expense of the protective alkali. These results are shown by the curves in figure 40. No remark need be made other than that needed to call attention to the rate of precipitation, which was 93.5 per cent in one hour, whereas the capacity was at the rate of 144 tons a day. Furthermore, at the rate of 72 tons a day, 99.5 per cent precipitation was obtained.

SIMPLE PERVIOUS CATHODES WITH SILVER CYANIDE AND LIME.

A test was made in which there were employed six simple pervious charcoal cathodes and six simple pervious Acheson-graphite comb

anodes as electrodes; 20 liters of solution containing 20 grams KCy , 9.52 grams CaO or 0.476 per cent CaO , 38.714 grams silver in the form of KAgCy_2 , and current gaps of one-fourth inch and a voltage

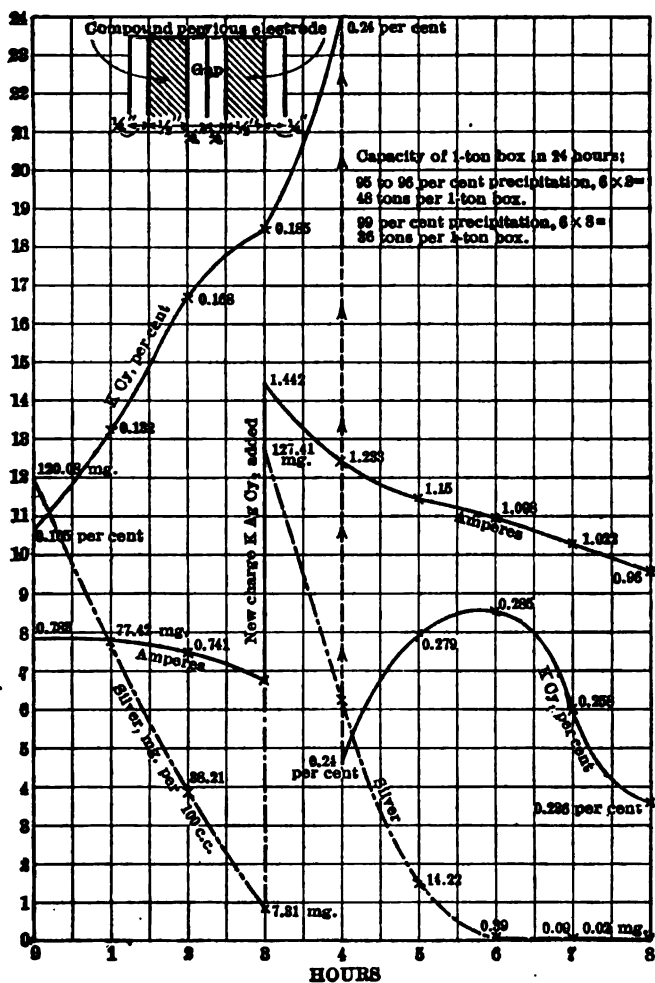


FIGURE 38.—Electrodeposition results with rich silver solution, using $\frac{1}{4}$ -inch current gaps. Eight compound pervious charcoal electrodes, each one-half inch thick; length of box occupied, $9\frac{1}{2}$ inches; cross section, 1 square decimeter. Eight current gaps at $3\frac{1}{2}$ volts, 28 volts in all. One Acheson-graphite comb anode. Solution, 20 liters, contained 20 grams KCy , 20 grams KHO , and 24.016 grams Ag as KAgCy_2 . Rate of flow not recorded; high. Note gain in KCy while Ag is being precipitated; loss afterwards

of $3\frac{1}{2}$. The cross section of the box containing the electrodes was 1 square decimeter, and the total length was $9\frac{1}{2}$ inches.

On plotting the results it was found that the amount of silver in solution closely followed the protective alkalinity. Thus as the

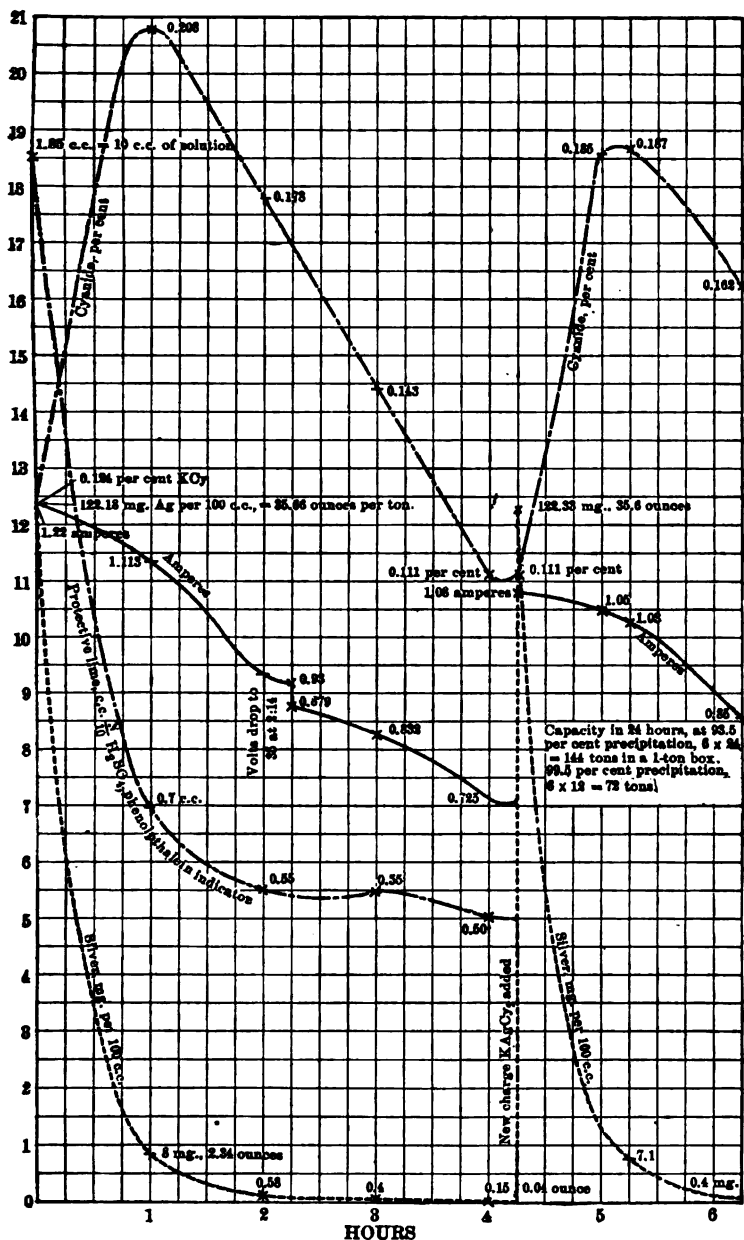


FIGURE 40.—Electrodeposition results showing effect of lime on the precipitation. One simple and 10 compound pervious electrodes, each one-half inch thick and containing $9\frac{1}{2}$ grams charcoal, 104 grams in all. Eleven one-fourth-inch current gaps at $3\frac{1}{2}$ volts, $38\frac{1}{2}$ volts in all. Solution, 20 liters, contained 20 grams K_2CO_3 , 10.2 grams (0.051 per cent) CaO , and 24.43 grams of silver as KAg_2O_3 ; 24.43 grams of silver as KAg_2O_3 was added at $4\frac{1}{2}$ hours; no $NaHO$ used. Rate of flow, 5 liters per minute; velocity, about 19.7 inches per minute.

PRECIPITATION OF GOLD FOLLOWING THE PRECIPITATION OF SILVER.

An interesting experiment was made in which two successive charges, the first of silver cyanide and the second of gold cyanide, were treated in the same box. The electrodes were already coated with silver and therefore were in excellent condition for conducting the current and depositing the gold, which was added from a cyanide solution containing \$24.36 per ton. In one hour 98.27 per cent of the gold was precipitated, the capacity for a 1-ton box being at the rate of 112 tons per 24 hours. The results are shown in Plate V.

It is evident from the precipitation curves that no good purpose was served in continuing the precipitation more than two hours, for when this was done cyanide was lost and little gold precipitated. The precipitation curves are found to be all logarithmic ones, and the precipitation extremely rapid at first, but slow toward the last. The economical period of the precipitation is therefore the first few hours, and this factor should be made use of to the fullest extent possible.

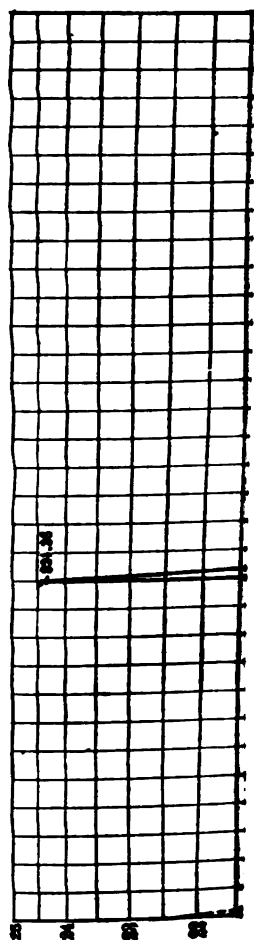
In conducting the experiment one simple pervious charcoal anode with graphite rod was used and 18 compound pervious electrodes, each containing 10 grams of excelsior charcoal. A cathode of the simple-pervious type containing 8 grams of excelsior charcoal was used. There were 14 current gaps, each one-fourth inch. The voltage between electrodes was $3\frac{1}{2}$ and the total voltage 49. The electrodes were 1 square decimeter in area and the length of the box 13 inches. The solution, 20 liters in volume, contained 20 grams NaHO, 20 grams KCy, 24.07 grams silver as KAgCy₂. The rate of flow was 13.8 inches per minute.

At the end of one hour 87.2 per cent of the silver was precipitated. Hence the capacity of the box for a 1-hour treatment is 4.67 times 24, or 112 tons for a 1-ton box in 24 hours. In 1 hour and 30 minutes there was 98.70 per cent precipitated; the capacity was 4.67 times 18 or 84 tons for a 1-ton box. In two hours 99.78 per cent was precipitated, hence the capacity for a 2-hour treatment was 4.67 times 12, or 56 tons for a 1-ton box.

At the end of this experiment 0.808 gram of gold in the form of KAuCy₂ and 0.685 gram of silver as KAgCy₂ were added to the solution without removing the electrodes and the current was started as before. The remarkably good results shown for gold are probably due to the fact that the cathodes were already coated thoroughly with metallic silver, which made them an excellent conductor and facilitated the precipitation of the gold.

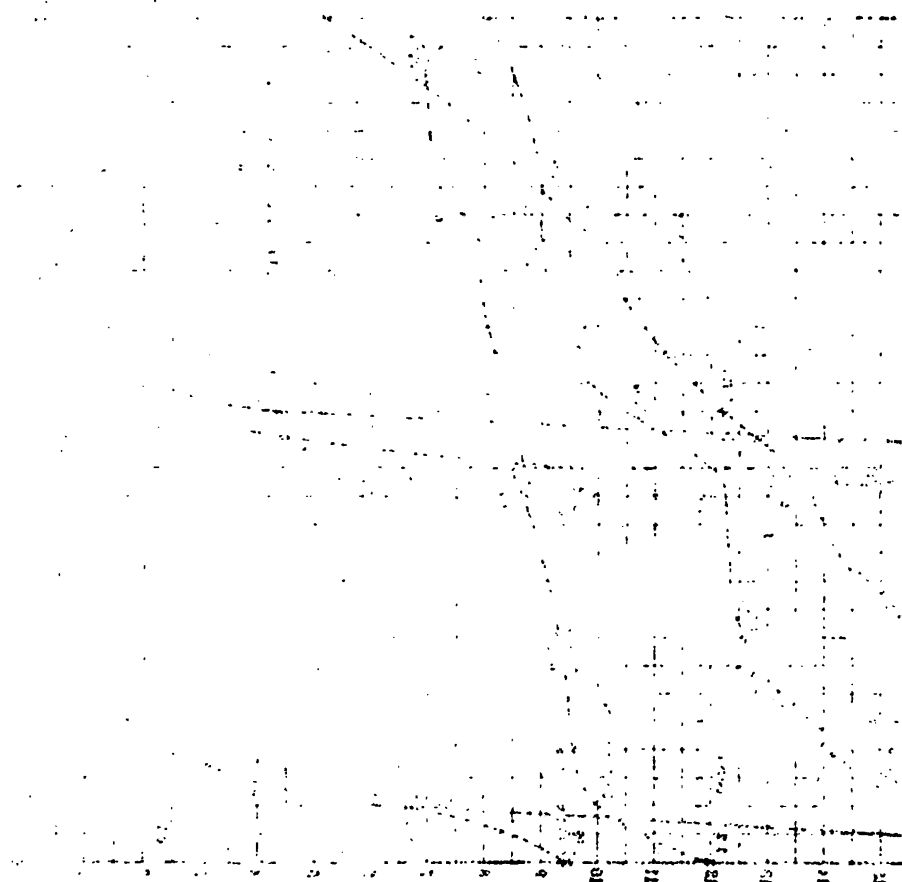
In the first hour there was precipitated 98.27 per cent of all the gold present. Hence the capacity of the box, which contains 4.28

BUREAU OF MINES



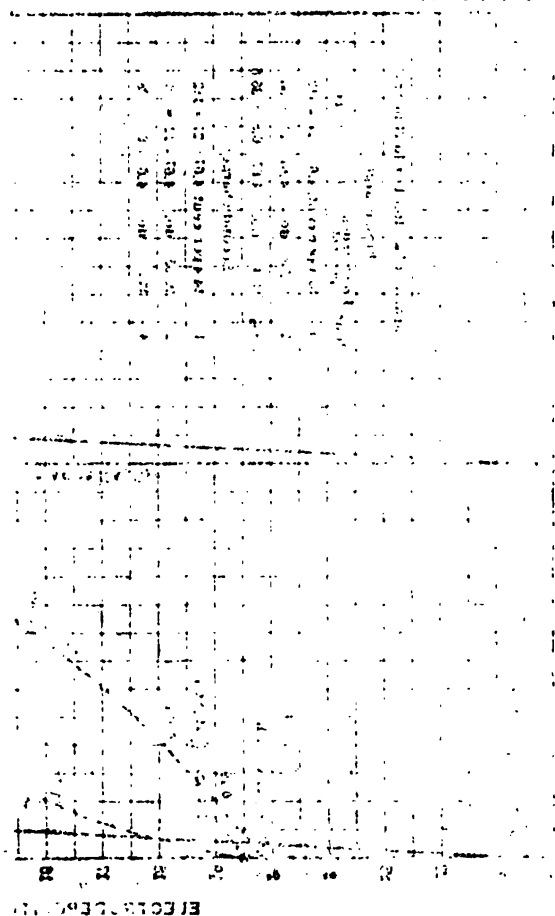
One Acheson graph
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NO RESULTS WITH CAUSTIC LINE IN 1 HOUR 14.

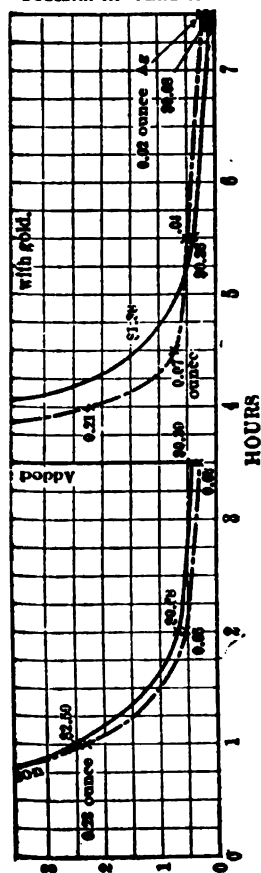
1. The first step in the process is to identify the problem or issue that needs to be addressed. This involves gathering information and understanding the context of the problem.



ELECTROLYTE

is shown in the graph. The curves show the effect of the concentration of the electrolyte on the rate of deposition. The curves are labeled with values like 0.25, 0.20, 0.15, 0.10, 0.05, and 0.02.

BULLETIN 100 PLATE VI



int gaps at 3 1/2 volts each,
KAgCy₂, and no KHO or

liters, would be 4.67 times 24, or 112.08, tons for a 1-ton box in 24 hours. At the end of the first two hours 99.75 per cent of the gold was precipitated, making the capacity of the box equal to 4.67 times 12, or 56.04, tons for a 1-ton box in 24 hours. It will be noticed that there was no advantage in running after the first two hours, as the amount of gold left at the end of that time was only 6 cents per ton. This content was reduced in the full three hours to $1\frac{1}{2}$ cents, but it was accomplished at the cost of destroying the total remaining cyanide. The desirable point at which to stop the precipitation is thus seen to have been probably after the first or second hour.

THE DIRECT USE OF A 110-VOLT CIRCUIT.

An experiment was undertaken to illustrate a mode of utilizing a 110-volt circuit, in which, for the purpose, two 13-inch boxes such as I have described were placed in series. These were given, in the first part of the experiment, a total tension of 98 volts, having altogether 28 current gaps at $3\frac{1}{2}$ volts each. The result, as regards precipitation, was very satisfactory. During the first half hour there was precipitated 93 per cent of the silver, giving a capacity of 112 tons for a 1-ton box. At the end of the first hour 99.8 per cent was precipitated.

The positive current was introduced by a simple pervious anode. Its course was through 13 compound pervious electrodes, out through a simple pervious cathode to a simple pervious anode at the head of the second box, and thence through 13 compound pervious electrodes to a simple pervious cathode. To adjust the voltage a resistance similar to that shown in Plate IV, A, containing twenty 32-candlepower incandescent lamps connected in parallel, was put in series with the cells. As there were 28 current gaps in the two boxes at 3.5 volts, the total voltage was 98 volts.

In starting the experiment, all of the twenty 32-candlepower lamps, connected in parallel, were placed in series behind the box. This gave at the start 98 volts. First one and then a second of these lamps, and so on, were disconnected, to adjust the voltage and to produce the effect wanted. The total length of electrodes in the two boxes was 24 inches; the current gaps were each one-fourth inch. Twenty liters of solution, containing 20 grams KCy and 20 of KHO, was used. The results are shown in figure 41.

In the first half hour, at 98 volts, 93 per cent of the silver was precipitated, the capacity being equivalent to 112 tons for a 1-ton box for 24 hours. At the end of the first hour, at 98 volts, 99.8 per cent was precipitated, the corresponding capacity being 56 tons for a 1-ton box in 24 hours. During the second hour, at 60 volts, 98 per cent was precipitated, and the capacity would be 56 tons for a 1-ton box. The

rate of precipitation, however, had nearly reached the maximum, with the 60-volt charge, at the end of the fourth hour, then fell slightly below 96 per cent in one hour, but at the end of the sixth hour was 99.85 per cent.

PRECIPITATION OF RICH GOLD AND SILVER SOLUTIONS IN THE PRESENCE OF LIME.

An experiment was undertaken to determine the effect of caustic lime (CaO) on the precipitation both of gold and silver from strong cyanide solutions. The results are shown in Plate VI.

The solution contained about $1\frac{1}{4}$ ounces of gold and 1 ounce of silver. A single box with 14 one-fourth inch current gaps was used. The results are to be compared with those shown in figure 41, where the precipitation was more rapid on account of the electrodes having been previously coated with silver and on account of the absence of lime.

In treating such rich solutions there is no necessity of reducing the gold content much lower than 90 per cent, as the solution is used over and over again. The presence of a small amount of gold in the leaching solution has no bad effect on the extraction. The most economical results are obtained in that way. A careful examination of the curves shows the point at which to stop in any given experiment.

The 13-inch box contained 1 simple pervious anode, 13 compound pervious electrodes, and 1 simple pervious cathode. Each of the 15 electrode boxes contained 10 grams of excelsior charcoal. The total length of electrodes was 12 inches, the cross section being 1 square decimeter, or one-ninth square foot. The 14 current gaps, of one-fourth inch each, at 3.5 volts, required 49 volts. Twenty liters of solution were treated, containing 40 grams of KCy , 0.808 gram of gold as KAuCy_2 , 0.636 gram of silver as KAgCy_2 , and 10.92 grams of CaO .

During the first hour 89.4 per cent of the gold was precipitated, in which time the box treated 4.67 times its content of solution, making the capacity equivalent to 112 tons in 24 hours for a 1-ton box. During the second hour 97.3 per cent of the gold was precipitated, giving a corresponding capacity of 56 tons. At the end of three and one-half hours 98.4 per cent of the gold was precipitated, the capacity corresponding to this rate being 36.5 tons. The second charge was treated four hours, in which time the gold precipitation was 99.5 per cent. The corresponding capacity of a 1-ton box would be 28 tons of solution in 24 hours.

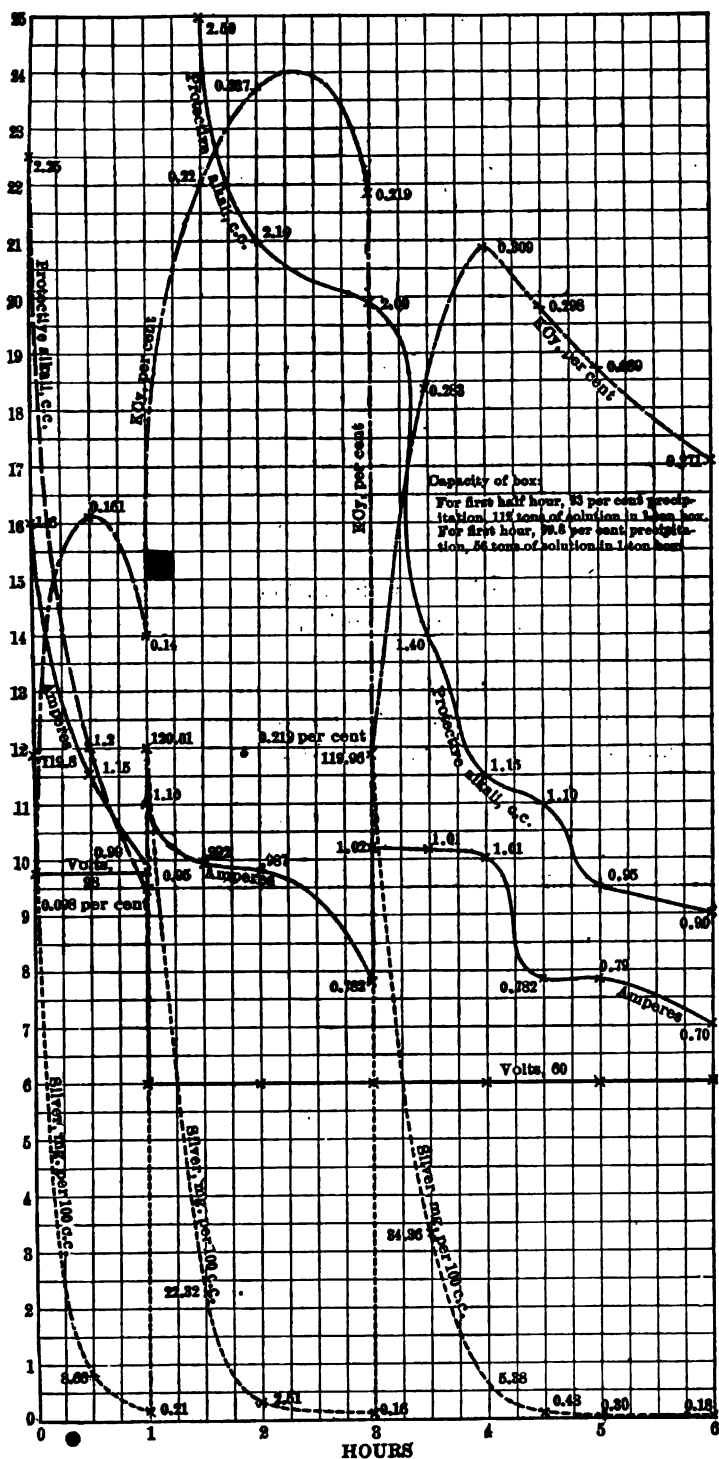


FIGURE 41.—Electrodeposition results with 110-volt direct-current circuit as source of power. Two electrode boxes in series, each having 1 simple previous anode and 18 compound previous cathode. Voltage regulated by bank of lamps. Twenty-eight current gaps: mean current drop from box to box for first hour, 98.8 volts; for rest of test 60–28=2.22 volts. Silver solution, 20 liters, contained 20 grams KCl and 20 grams KHO. Rate of flow not recorded.

EXAMPLE OF CLEAN-UP FROM SILVER SOLUTIONS.

Three charges of silver of 23.59 grams each in the form of KAgCy_2 , making a total charge of 70.77 grams of silver, were run through the deposition box. There were used 1 simple pervious anode of excelsior charcoal, 13 compound pervious electrodes, and 1 simple pervious cathode, each containing 8 grams of charcoal.

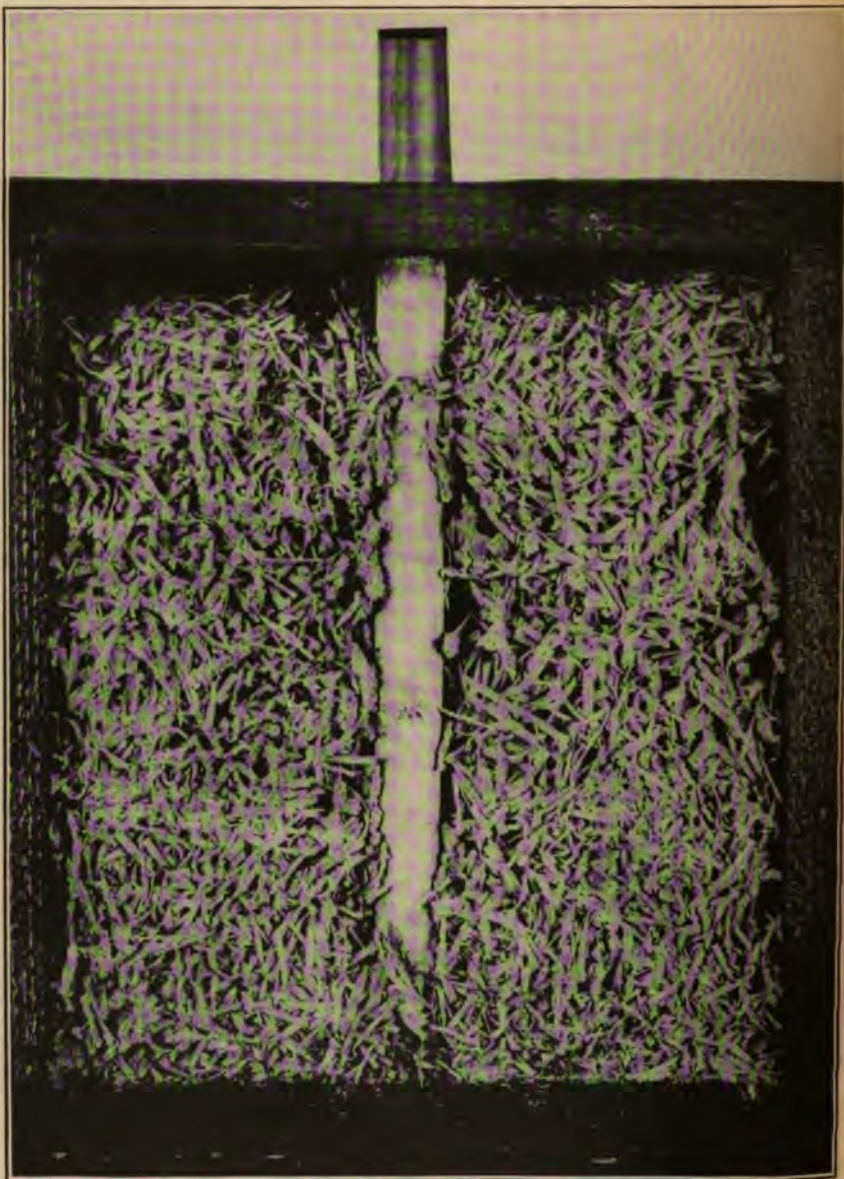
The solution after treatment contained 0.63 mg. silver per 100 c. c., there being left in the 20 liters of solution 0.126 grams. The charcoal of the cathode was thoroughly gorged with silver. The upstream or cover side of the cathode is shown in Plate IV, *B* (p. 143), the cheesecloth being covered with a mass of pure crystals of silver. The deposit had so extended that there was danger of short-circuiting.

Plate VII shows the reverse side of this cover. Here the cheesecloth is seen to support a mass of silver crystals, together with fragments of excelsior charcoal, coated with silver. The silver on this cover was very easily removed by scraping and brushing under water.

Plate VIII shows the interior of the cathode box. The graphite conducting rod is seen to be covered with a loosely adhering coating of silver crystals. The charcoal in the box is seen to be thoroughly coated with silver, the weight of silver being more than three times that of the charcoal. The charcoal in the compound pervious electrodes, as usual, was not so densely plated as at the cathode at the end of the box. The total result of the clean-up, consisting of 70.644 grams and 4.956 grams of ash, was as follows: From the simple anode, 25.3 grams; from the 14 compound pervious electrodes, cathode side, 46.2 grams; from the 14 anode sides, 4.1 grams.



BACK OF COVER. SIMPLE PERVIOUS CATHODE NEARLY CHOKED WITH SILVER.



SIMPLE PERVIOUS CATHODE USED AT END OF COMPOUND PERVIOUS ELECTRODE; 25.3 GRAMS OF SILVER DEPOSITED ON 8-GRAM MASS OF EXCELSIOR CHARCOAL.

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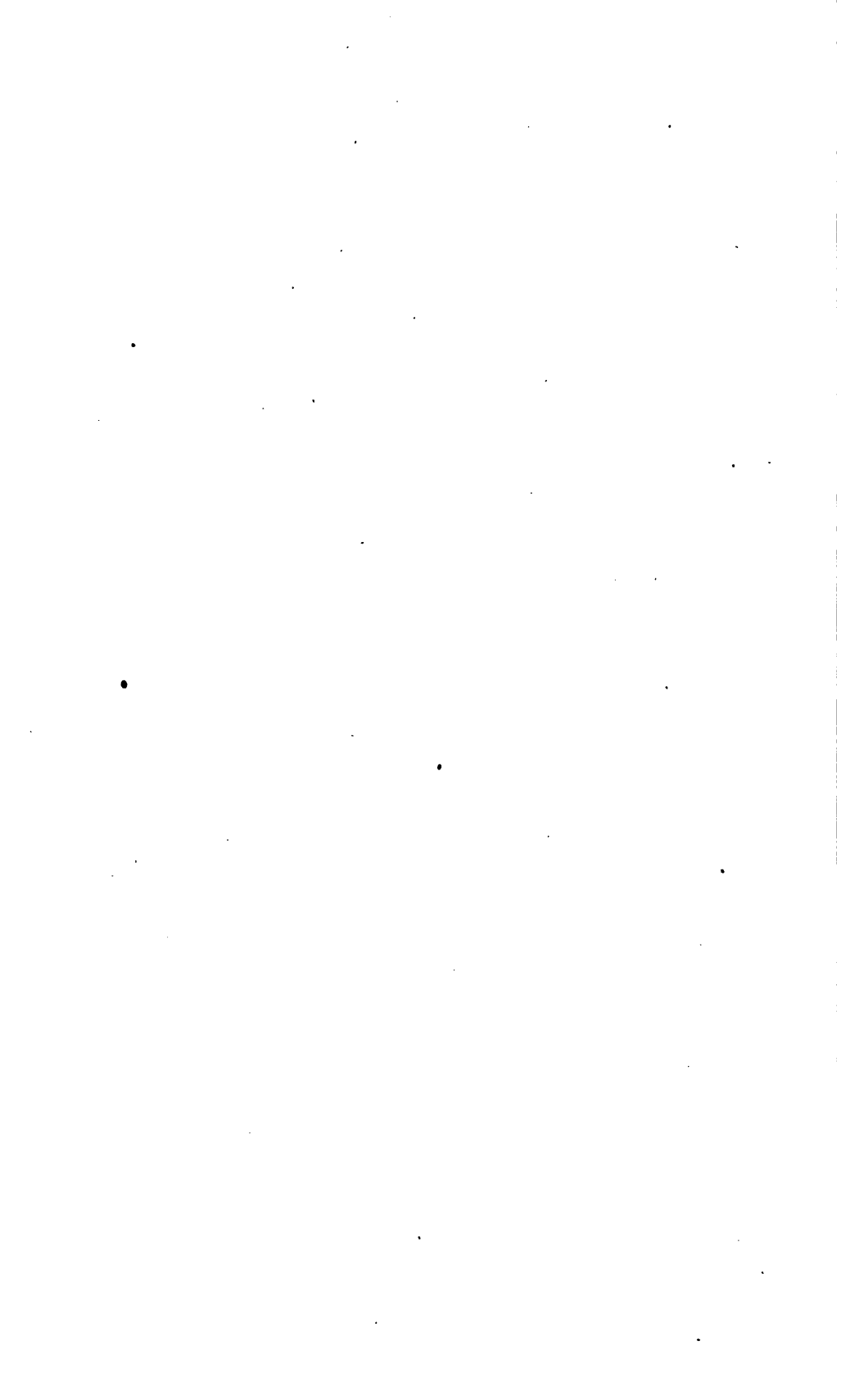
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DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

**RECOVERY OF GASOLINE
FROM NATURAL GAS BY COMPRESSION
AND REFRIGERATION**

BY

W. P. DYKEMA



**WASHINGTON
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RECOVERY OF GASOLINE FROM NATURAL GAS BY COMPRESSION AND REFRIGERATION.

By W. P. DYKEMA.

INTRODUCTION.

In investigating the general problems that relate to the petroleum industry in the United States, the Bureau of Mines has given considerable attention to the recovery of motor fuel from natural gas. Recent developments in gasoline power units and their increasing use have made it imperative that all fractions of petroleum suitable for fuel in this type of engine be conserved. The bureau has issued a number of publications on this subject. Among these are Technical Paper 10, "Liquefield products of natural gas, their properties and uses"; Bulletin 42, "The sampling and examination of mine gases and natural gas"; Technical Paper 87, "Methods of testing natural gas for gasoline content"; Bulletin 120, "Extraction of gasoline from natural gas by the absorption method"; and Bulletin 88, "The condensation of gasoline from natural gas."

This report treats of the compression and refrigeration process for the recovery of gasoline from natural gas from the viewpoint of the practical engineer and business man. Conditions of actual operation and the equipment in use are cited and described so that operators, and others interested, can compare the variations in methods of treating natural gas for its gasoline content in the different fields and also the conditions encountered and the features that control the methods used.

ACKNOWLEDGMENTS.

The writer heartily acknowledges the valuable assistance and cooperation of the many plant operators and gasoline producers who have cheerfully furnished data on their plants and granted the privilege of plant inspection.

Particular thanks are due to Messrs. W. R. Hamilton, G. L. Goodwin, and D. L. Newton for the many courtesies extended the writer and for much of the information regarding cooling by expanded gas as practiced in the California fields.

Thanks are also due Messrs. W. P. Donovan, Thomas Chestnut, J. C. Gillespie, A. W. Peake, and R. E. Downing for valuable information on practices in the Mid-Continent fields.

HISTORY OF THE INDUSTRY.

The early history of the manufacture of natural-gas gasoline has been published by the Bureau of Mines ^a in previous papers on the subject. The present practice is the result of the advance developed by a study of the needs of the industry, by improvements in machinery and equipment, and by a better understanding of the thermodynamic and other physical principles involved. A very small proportion of the plants erected have been commercial failures, and most of those which have failed lacked capital to carry them over a period of low prices for gasoline. Many of these plants are still operating, the company having been reorganized or bought up by larger corporations, and under present market conditions they show satisfactory returns.

The following table shows the distribution of casing-head gasoline plants and the amount and value of the gasoline produced in the United States in 1915:

TABLE 1.—Gasoline recovered from natural gas and sold in 1915.^a

State.	Number of plants.	Quantity.	Value.	Average recovery of gasoline per 1,000 cubic feet.
		<i>Gallons.</i>		<i>Gallons.</i>
Oklahoma.....	63	31,665,991	\$2,361,029	3.60
California.....	20	12,835,126	975,397	1.60
West Virginia.....	114	10,853,608	927,079	2.30
Pennsylvania.....	139	5,898,597	569,873	2.73
Ohio.....	50	2,198,715	167,138	2.80
Illinois.....	16	1,035,204	80,049	2.29
Texas.....	1			
New York.....	4			
Louisiana.....	2			
Kansas.....	2			
Colorado.....	2			
Kentucky.....	1			
	414	65,364,665	5,150,823	2.57

^a Northrop, J. D., Mineral Resources U. S. for 1915, U. S. Geol. Survey, 1916, p. 997.

^b Includes gasoline resulting from natural condensation in gas mains.

In January, 1917, the number of plants in California had increased to 30, with an estimated daily production of more than 60,000 gallons, and the number of plants in Oklahoma to 95, with an estimated daily production of 200,000 gallons.

^a Allen, I. C., and Burrell, G. A., Liquefied products from natural gas, their properties and uses: Tech. Paper 10, 1912, pp. 4-7; Burrell, G. A., The suitability of natural gas for making gasoline: Tech. Paper 57, 1913, p. 17; Burrell, G. A., Selbert, F. M., and Oberfell, G. G., The condensation of gasoline from natural gas: Bull. 88, 1915, pp. 9, 10; Burrell, G. A., Biddison, P. M., and Oberfell, G. G., Extraction of gasoline from natural gas by absorption methods: Bull. 120, 1917, pp. 11-14.

FACTORS TO BE CONSIDERED IN EXAMINING FIELD FROM WHICH GAS IS TO BE TAKEN.

Before a compression plant can be properly designed to treat natural gas from any given area, a thorough study of the history of the field in general and especially of the sand from which the gas is to be taken should be made, also complete testing to determine the volume and gasoline content of the gas from the area under consideration should be made. This precaution is particularly necessary for a field in which there are no plants in operation that can be studied and used as a precedent.

A knowledge of the history of the field is important in determining the probable life of the wells and the decline in volume of gas produced from year to year. A plant built in California to treat 2,000,000 cubic feet per day has been able to get only half that amount because of the decline in gas production from the wells after the initial rock pressure was relieved, and new wells drilled for oil on the same lease have not thus far brought the quantity of gas up to the plant capacity. Gas from adjoining leases could not be obtained for treatment, and the plant at the time of the writer's visit was running at half capacity, as stated.

In all gas and oil fields in the United States it is found that as the pressure in the wells decreases the gas becomes richer in gasoline content. It is doubtful, however, if the actual total quantity of gasoline vapor from a given well increases, but it seems to be a fact that the amount of gasoline vapors produced declines much less rapidly than either the oil or gas production from a given well or area. This has been found to be practically true of West Virginia and California wells as long as the wells produce oil either under rock pressure or vacuum. In Pennsylvania a small compression plant is treating gas from wells that have long ceased to produce oil and are pumping salt water under vacuums of 26 inches of mercury.

New oil wells producing many million feet of gas under high rock pressure, such as are often brought in, in the Mid-Continent and California fields, are not considered, because gas produced under these conditions of pressure and volume practically never finds its way to compression plants. The gas produced under rock pressures of 400 to 1,500 pounds per square inch contains only comparatively small proportions of the heavier gasoline fractions and comparatively large quantities of the lighter or "wild" fractions, as would be expected from the condition under which the gas is held while in contact with the oil from which it must receive its charge of condensable vapors.

CHARACTER OF THE SAND.

A knowledge of the sand from which the gas is to be produced is a valuable factor in designing a plant; the thickness, texture, or degree of cementation bear directly on the future of the field as a

gas and gasoline producer. Thick sand of close texture indicates long life as an oil producer, and consequently a long life for gasoline production. Loose, uncemented sands, such as are found in California, will not withstand the suction of vacuum pumps, which causes the sand to come in and stop or injure the pumps. This inflow of sand reduces the production of oil and increases the expense of cleaning the wells, so that companies treating their own gas do not permit the vacuums held on walls to exceed 1 to 2 inches of mercury at the casing head. This limitation of vacuums may react in such a way as to make compression plants unprofitable when the oil production has become small and the rock pressure has been completely relieved.

CHARACTER OF THE OIL.

Casing-head gas produced with high-gravity oil is almost universally rich in gasoline vapors, except when the gas is produced under extremely high pressures. However, the fact that an oil is of low gravity can not always be depended on to indicate small gasoline content, as has been shown by compression practice in California. As a rule, in California, casing-head gas produced with oil having a gravity of less than 22° B. does not yield enough gasoline for profitable compression. However, in the Salt Lake field, the oil has a gravity of 15° to 17° B. and still the casing-head gas is commercially valuable for its gasoline content. Analyses by the Bureau of Mines^a of oil from this field shows that it carries exceptionally high proportions of asphaltum (55.3 per cent), but also carries lighter products, ranging up to fractions distilling over at 150° to 200° C., to which are due the gasoline vapors in the gas. The Kern River field, yielding oil with an average gravity of 14° B. has not, so far, produced any gas that is being treated for gasoline.

WATER SUPPLY.

Water being essential in all plants making gasoline by compression methods, the supply and the quality of water available should be carefully determined. The cooling coils and the compressor jackets of a plant treating 1,000,000 cubic feet of gas daily at a pressure of 250 pounds per square inch will use 100 to 300 barrels of water a day, depending on the design of the water-cooling system and the temperatures obtained. The loss is accounted for by the evaporation and, in plants where towers or sprays over ponds are used, by water being carried away by the wind. In the oil fields much of the water is so heavily charged with mineral salts as to be almost useless for boilers, gas-engine jackets, and compressor jackets. Such water to be made fit for boiler use must be treated with a so-called "boiler

^a Allen, I. C., Crossfield, A. S., Jacobs, W. A., and Matthews, R. R., *Physical and chemical properties of the petroleum of California*: Tech. Paper 74, Bureau of Mines, 1914, p. 13.

compound." For cooling jackets of machines the water must be condensed, otherwise it forms a scale, which interrupts the circulation of water and is dangerous both to machinery and to the operators, and also cuts down the efficiency of the engine and compressor by permitting overheating of the cylinders and of the gas being treated.

TRANSPORTATION FACILITIES.

The matter of transportation is of importance, as on it depends, in part, the questions of blending, size and weight of units, plant location, and length of pipe lines. Before a plant is designed, the most economical product should be determined, as it may be that producing the maximum quantity of condensate possible from a given gas will be as profitable as a smaller quantity of a less volatile product. Producing the maximum quantity of condensate from a gas requires high pressures and blending with naphtha as soon as possible after the condensate is precipitated. This requires machines to produce the high pressure, which adds to the cost of installation, and a continual supply of naphtha for blending. If the cost of freight on the naphtha coming to the plant and on the percentage of naphtha in the blended product going back to market, plus the cost of mixing and of losses in blending and handling, is greater than the value of the condensate produced in excess of the quantity that could be produced at a lower pressure without blending, it would be more profitable to produce less condensate of lower vapor tension capable of being shipped as made. For example, a plant in the Mid-Continent field, situated an excessive distance from the nearest refinery from which it could obtain naphtha for blending, finds that using a pressure of 80 pounds from single-stage compressors, and producing 1.8 gallons of condensate with a gravity of 75° to 80° B. and a vapor tension of 5 to 6 pounds, is more profitable than producing 3 gallons of condensate having a gravity of 94° to 98° B. and blending it at or near the plant.

The distance from a railroad will also have a bearing on the cost of erecting a plant. Pipe lines are usually built from the plants to loading racks or blending stations on the railroad. More recent practice seems to favor blending at the compression plant, which necessitates pumping the naphtha to the plant and pumping the blended product back to the loading station through pipe lines. The greatest expense from being at a distance from a railroad, however, is that of hauling equipment and repairs. Machines built in parts too large to handle on the wagons or trucks usually found in oil fields are much more expensive to place than machines shipped in smaller parts. Roads, distances, and the weights of large parts of machines should be considered when a plant is being designed or an estimate of cost figured.

Plate I, A (p. 24), shows two single castings weighing 31,000 pounds each, which were placed in a plant that could be reached only by narrow, steep roads. The trouble and expense of hauling such parts is a factor in plant construction.

The general topography at one compression plant is shown in Plate II, A (p. 24).

DISTRIBUTION OF WELLS.

If the wells from which gas is to be taken are distributed over a wide area, it is well to consider the construction of two or more smaller plants rather than one larger plant. One plant visited by the writer treats gas collected over an area of approximately 32 square miles. The extensive pipe-line system required to bring the gas to the plant and return it to the various leases, with the booster stations and equipment necessary to maintain them, makes this plant, in the opinion of the management, more expensive and less efficient than two or three smaller plants would have been.

CONDITION OF WELLS.

Before erecting a compression plant in an old field the condition of the casings in the wells from which gas is to be taken should be ascertained. In an old field in Pennsylvania it was found that the casings were badly rusted and allowed excessive quantities of air to enter the lines as soon as a vacuum of more than 1 or 2 inches was placed on them. The wells made an average production of only 1,000 cubic feet of gas a day, so that the expense of recasing would not seem to be warranted.

J. O. Lewis, petroleum technologist of the Bureau of Mines, states that air is often admitted to wells in which the casing has not been properly placed, the air entering through porous strata from other wells which are not being held under vacuums.

TESTING NATURAL GAS.

The testing, measuring, and sampling of natural gas for its gasoline content has been described in Bureau of Mines publications^a by G. A. Burrell and his associates, and will only be taken up briefly by the writer.

Many compression plants are successfully treating gas about which few facts were known before the plant was erected. Much time and expense could have been saved, however, if more had been learned of the physical and chemical characteristics of the gas before the plans

^a Burrell, G. A., and Jones, G. W., Methods of testing natural gas for gasoline content: Tech. Paper 87, 1916, 23 pp. Burrell, G. A., Selbert, F. M., and Robertson, I. W.: Analysis of natural gas and illuminating gas by fractional distillation at low temperatures and pressures: Tech. Paper 104, 1915, 41 pp. Burrell, G. A., Selbert, F. M., and Oberfell, G. G., The condensation of gasoline from natural gas: Bull. 88, 1915, 106 pp.

of the plant were made and the machinery installed. The investment in a compression plant is large enough to warrant the expense of having all necessary tests, analyses, and measurements of gas made and checked before the plans and details of construction are taken up.

SPECIFIC GRAVITY TEST.

Natural gas having a specific gravity of 0.78 (air = 1) and higher is being successfully treated in compression plants. The specific gravity test is useful as an indicator, but the possibility of misleading variations through the presence of other gases, such as air, carbon dioxide, nitrogen and sulphur compounds, in the sample makes this test unreliable if used alone. If an analysis is made of the gas and the specific gravity of the hydrocarbon contents computed, the results are more dependable, but even then are not reliable enough to be used as a basis for final decisions regarding plant construction.

SOLUBILITY TEST.

The solubility test described by Burrell and Jones^a is useful as an arbitrary test, because it is known that gases of a solubility of less than 30 per cent have not been successfully treated in compression plants.

In regard to the following table they state that "in all cases the yield represents the actual amount of gasoline sold after weathering."

Yield of gasoline from casing-head natural gas by compression method, corresponding to absorption and specific-gravity tests.^a

Absorption by oil.	Specific gravity (air=1).	Yield of gasoline, gallons per 1,000 cubic feet of gas.	Absorption by oil.	Specific gravity (air=1).	Yield of gasoline, gallons per 1,000 cubic feet of gas.
<i>Per cent.</i>			<i>Per cent.</i>		
16	0.64	None.	50	1.29	3.00
22	0.83	1.00	48	1.37	3.50
30	0.90	1.75	44	1.38	3.50
37	1.00	2.00	65	1.38	4.00
39	1.08	2.50	84	1.41	4.50
38	1.07	3.00	86	1.46	5.00
54	1.21	3.50			

^a Burrell, G. A., and Jones, G. W., work cited, p. 10.

It should be stated that both the specific-gravity test and the oil-absorption test fail when applied to residual gas from a gasoline plant because, although the results obtained will indicate high specific gravity and oil absorption, principally because of the presence of large percentages of the hydrocarbon gases, ethane and propane, yet the plant will have extracted the vapors of the liquid paraffins that constitute gasoline.

^a Burrell, G. A., and Jones, G. W., Methods of testing natural gas for gasoline content: Tech. Paper 87, Bureau of Mines, 1916, pp. 7-10.

FRACTIONAL DISTILLATION.

The Bureau of Mines ^a has developed a technical laboratory test based on a method of freezing out the propane and higher hydrocarbon fractions, which accurately determines the percentage of condensable hydrocarbons, including propane and all other higher members of the hydrocarbon series. This test can be made only in a well-equipped laboratory by experienced gas analysts.

Another method, based on the principle of low temperatures, that was originated by the Smith Emery Co., of Los Angeles, Cal., is essentially as follows: Gas from the casing-head of the well is led through a service meter (see fig. 1) under a pressure of 12 inches of water (devised so as to blow out if that pressure is exceeded) to a copper tube three-fourths of an inch in diameter and 15 feet long, coiled so as to fit into an asbestos insulated container 18 inches deep and 12 inches square. The container is filled with acetone (product of wood distillation) to within 2 or 3 inches of the top. Carbon dioxide snow from steel bottles of carbon dioxide in the liquid state is added to the acetone until a saturated solution is obtained. Complete saturation of the solution is shown when a portion of the snow remains, as snow, on the bottom of the container. The temperature of the acetone is held constant at 70° F. below zero by the carbon dioxide used in this way. Gas is passed through the copper coil, submerged in the acetone bath, until 500 or 1,000 cubic feet, as shown by the meter, has been treated; the coil is then drained into a flask, the quantity of condensate measured, and the gravity tested. If the condensate obtained in this manner is higher in gravity and vapor tension than would be desirable as a plant product, the condensate can be weathered down to the desired product, thus indicating the quantity of condensate of desired gravity which the gas contains. One series of tests made in this manner produced a condensate having a gravity of 90° B., and an absorption plant treating the gas reports a production approximately equal to the results obtained in the original tests of the gas. It requires 2 or 3 bottles of carbon dioxide to conduct this method of testing for one day, and the gas from 6 to 10 wells can be tested in that length of time.

PORTABLE COMPRESSOR TEST.

It has become the practice, in testing natural gas for its gasoline content to determine its suitability for use in a compression plant, to make actual physical tests with a small portable plant consisting of a compressor, meter, and cooling coils mounted on a wagon or

^a Burrell, G. A., Seibert, F. M., and Robertson, I. W., *Analysis of natural gas and illuminating gas by fractional distillation in a vacuum at low temperatures and pressures*: Tech. Paper 104, Bureau of Mines, 1915, 41 pp.

truck. Each well is tested by moving the compressor outfit to a point where gas can be taken from the casing head or from the gas line leading from the well. The reason for testing each well separately is that even in the same field gas from different wells has widely varying contents of gasoline vapor, and if comparatively dry gas is mixed with gas of greater vapor content, the yield of condensate will be decreased or greater pressure and a lower temperature will be required to precipitate an equal quantity. This point was proved in practice by a plant in California, which, by turning out of its lines the gas from a well making about one-half million cubic feet per day, increased the plant production, no change being made in the pressure or the cooling system.

A portable testing outfit observed in operation by the writer consisted of a tank 12 by 36 inches used as a gas receiver, a 4-horsepower gas engine belted to a 3 by 3½ inch single-acting compressor with a capacity of 3 cubic feet per minute, a single coil of 1-inch pipe of the continuous, return-bend type cooled by submerging in a wooden trough of water and ice, and a double coil, 12 feet long, of 1-inch pipe inside a 2-inch pipe, cooled by expanding the compressed gas through a valve connection between the outside and inside pipes.

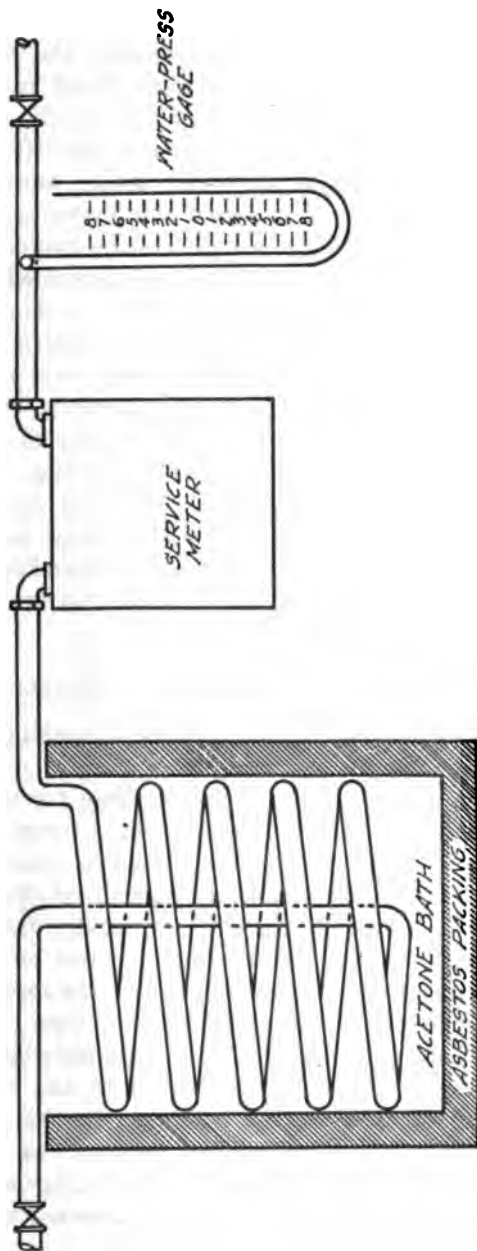


FIGURE 1.—Apparatus for testing natural gas by the acetone and carbon-dioxide method.

The gas was brought from the well line through a $\frac{1}{2}$ -inch pipe fitted with a water gage for regulating the pressure on the intake receiver; this receiver also served as a trap for heavy oil and dirt. From the receiver the gas was compressed and delivered to the water-cooled coil at a pressure of 250 pounds, the water being cooled with ice. Ice was used because it gave lower and easily regulated temperatures, was more convenient to obtain than a large or continuous supply of fresh cold water, and permitted tests to be run at nearly the same temperature and under more uniform conditions. The gas from the water-cooled coil discharged into the chamber between the outer and the inner pipe of the double coil. At the discharge end of the outer or 2-inch pipe was suspended a 1-inch drip pipe to collect the condensate. Gas from the 2-inch pipe was expanded through the valve into the inner pipe to refrigerate the high-pressure gas flowing through the outside pipe and discharged through a service meter to the atmosphere. Records were kept of temperatures, pressures, and amounts and gravity of condensate produced. The tests were made at night to aid cooling. From time to time specific-gravity tests of gas from individual wells were made and recorded. When the gas from any well dropped noticeably in gravity, a compression test was run on it, and, if found too low in gasoline content for profitable recovery, was turned into the fuel lines of the lease and was not treated.

SOURCES OF ERROR IN PORTABLE COMPRESSOR TESTS.

The greatest source of error in making tests with portable outfits is the meters. Service or domestic meters become inaccurate if operated at pressures above those for which they were made, and should be tested before use even at normal pressures, and during use should be protected from excessive pressures by a water or mercury pressure gage placed on the pipe ahead of the meter intake. Pressures of 8 to 12 inches of water are usually used with meters of this type. Some operators use two meters, one on the intake and one on the discharge of the portable tester, the indicated amounts of gas being averaged in calculating the production.

Results are apt to be misleading when gas being treated in the testing machine is not cooled to the same temperature as under plant conditions or compressed to the pressure used in the plant. Small portable testing compressors can be adjusted and operated under conditions so nearly simulating actual plant conditions as to give reliable data on which to base estimates of pressures, temperatures, and recovery.

MEASURING THE FLOW OF NATURAL GAS.

The following description of the orifice-meter method of gas measurement is from Technical Paper 87^a of the Bureau of Mines:

ORIFICE METER.

An instrument known as the orifice meter, for testing small flows of natural gas, is shown in figure 2. This instrument is simple in construction, consisting of a short 2-inch nipple, *b*, with pipe thread on one end and a thin plate disk on the other. The disk carried a 1-inch orifice, *a*, and a hose connection, *c*, for taking the pressure. The meter is especially intended for testing small gas wells and "casing-head" gas from oil wells. As a rule the flow of gas from an oil well is rather small, and it is not advisable to test the flow with a Pitot tube such as is used in testing large gas wells. In using the orifice tester it is necessary to know the specific gravity of the gas in order to obtain the flow.

Before the orifice well tester is attached to the casing head the well should be permitted to blow into the atmosphere until the head of the gas is reduced and the flow has become normal. Then the tester is attached by simply screwing it into the end of a 3-foot length of 2-inch pipe and the pressure is read in inches of water on the siphon gage, *d*. In the table^b the flow of the well, with values for gas of different gravities, is opposite the gage reading. The orifice in the instrument should be kept dry and uninjured; otherwise the gage reading will not be correct.

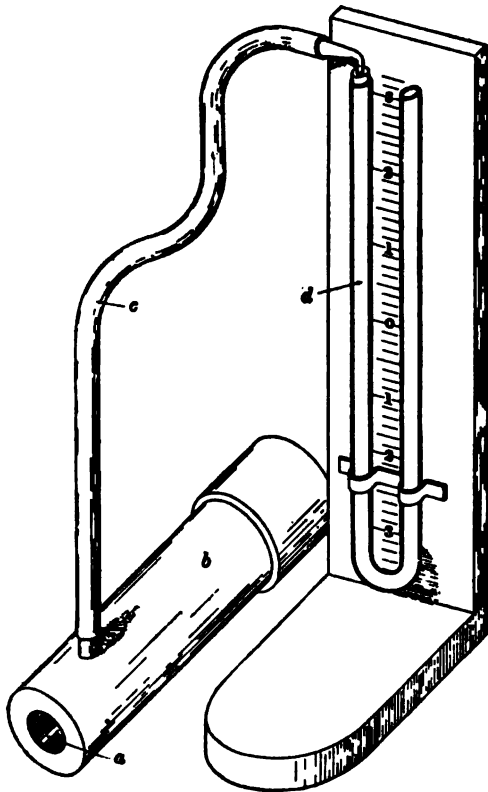


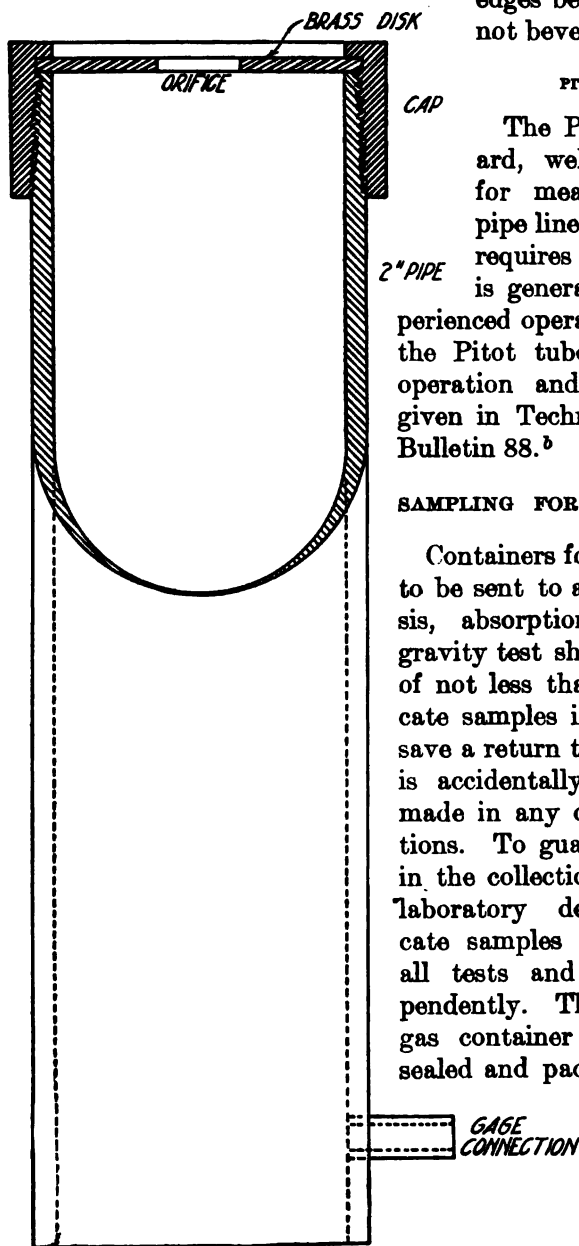
FIGURE 2.—Orifice meter for testing small flows of natural gas.

Beside the thin 1-inch orifice plate described above, orifice meters of this type are equipped with plates of the same thickness ($\frac{1}{8}$ inch) and with orifices of $\frac{3}{8}$, $\frac{1}{2}$, $\frac{3}{4}$, and $1\frac{1}{4}$ inch diameters for measuring flows of greater or less volume, and for checking, by two or more tests, the flow from the same well at different pressures. Figure 3 shows the design of meters in which are used a number of different sized orifice plates. The brass disks containing the orifices are carefully machined to a thickness of $\frac{1}{8}$ inch and are made to fit perfectly

^a Burrell, G. A., and Jones, G. W., *Methods of testing natural gas for gasoline content*: Bureau of Mines, 1916, pp. 18-20.

^b See Table 15, p. 115.

between the cap which holds the disk in place and the end of the 2-inch pipe. The orifice in the disk is accurately drilled to size, the edges being made square and not beveled.



PITOT-TUBE METHOD.

The Pitot tube is a standard, well-known instrument for measuring gas flows in pipe lines and at gas wells; it requires careful attention and is generally used only by experienced operators. Descriptions of the Pitot tube and its method of operation and tables of flow are given in Technical Paper 87^a and Bulletin 88.^b

SAMPLING FOR LABORATORY TESTS.

Containers for natural-gas samples to be sent to a laboratory for analysis, absorption test, and specific gravity test should have a capacity of not less than 1 pint, and duplicate samples in two containers may save a return to the field if a sample is accidentally lost or an error is made in any one of the determinations. To guard against errors both in the collection of samples and in laboratory determinations, duplicate samples are often taken and all tests and analyses run independently. The bottle or other gas container should be carefully sealed and packed to avoid leakage

from poor stoppers and expansion and contraction of the gas from wide changes in temperature.

FIGURE 3.—Orifice meter in which different sized orifice plates may be used.

^a Burrell, G. A., and Jones, G. W., Methods of testing natural gas for gasoline content: Bureau of Mines, 1916, pp. 21-24.

^b Burrell, G. A., Selbert, F. M., and Oberfell, G. G., The condensation of gasoline from natural gas: Bureau of Mines, 1915, pp. 37-45.

COLLECTING A SAMPLE BY AIR DISPLACEMENT.

Filling a bottle or other container by air displacement is a rapid and convenient method, often used in the field. A small hose is placed over a pet cock in the gas line, the other end being inserted inside of the container. A gentle stream of gas is then allowed to flow into the container for four or five minutes, displacing the air. If the gas is known to be lighter than air it is best to invert the bottle, thus allowing the light gas to displace the heavier air; if the gas is heavier than air, the reverse method is preferable. The bottle is then quickly corked, and sealed by covering the stopper with a coating of warm paraffin. The difficulty with this method is the uncertainty whether all of the air has been displaced by the gas.

COLLECTING A SAMPLE BY WATER OR MERCURY DISPLACEMENT.

A more accurate method of obtaining a gas sample is by the displacement of water. In this method the bottle is filled with water and inverted beneath the surface of water in a bucket or other convenient receptacle, the hose from the gas line is allowed to discharge below the mouth of the inverted bottle until the water in it has been entirely displaced by the gas. The bottle is then closed or corked while still beneath the surface, and after drying and sealing with paraffin, can be transported without danger of contamination of the contents.

Mercury may be used in the same manner as water, but requires that considerable volumes of this liquid be kept with the sampling apparatus, which adds to its bulk and weight. Displacement of mercury is, however, the most accurate method of filling a container with the gas to be tested, and insures the least possible contamination.

SAMPLING AND TESTING IN GENERAL.

All tests, measurements, and determinations of natural gas to be used for gasoline manufacture should be checked and proved. Duplicate tests, analyses, and measurements should be made by different persons, if possible. If the duplicate results show variations large enough to be of importance in determining plant capacity, methods, or equipment, a third test should be carried out to determine which results can be trusted and can be used in plant design. Too much importance can not be placed on accurate testing and measuring of gas to be treated for gasoline content, for although during the treatment in a commercial plant many problems arise that can not be foretold, many characteristics of the gas can be determined which have a direct bearing on the pressures and temperatures necessary and the gravity and vapor tension of the condensates produced.

METHODS OF NATURAL-GAS GASOLINE MANUFACTURE.

Plants treating natural gas for its gasoline content fall under one of three mechanical and physical subdivisions—compression, refrigeration, and absorption, or a combination of these methods. Each division varies widely in mechanical details of both equipment and operation. Gasoline recovery by compression and refrigeration will be discussed in the following pages. Recovery by absorption methods has been described in Bulletin 120.^a

THEORY OF PRECIPITATION.

CONDENSATION BY COOLING.

The theory of the precipitation of vapors from natural gas is in a way comparable to that of the precipitation of water vapor from the atmosphere. Water vapor is known to exist in the air at all times in varying proportions, but is invisible unless condensed by cooling. Cooling to a temperature below that at which the air is saturated with the water vapor it contains causes precipitation in the form of snow, hail, frost, rain, fog, or dew. This fact is also illustrated by beads of water collecting on the outside of a pitcher of ice water, and by the formation of frost on the refrigeration pipes of expansion units, as shown in Plates V, C, VI, and VII (pp. 34 and 36). The air coming in contact with the cold surface of the pitcher or pipes is cooled below its dew point and the moisture which it can no longer hold as vapor condenses as water or frost on the surface. By further cooling of the air more moisture would be condensed, and if cooling were carried far enough practically all of the water vapor could be precipitated without the aid of pressures higher than atmospheric.

CONDENSATION BY COMPRESSION.

In plants compressing air it is found that air, after having been made more dense by increase of the pressure and decrease of the volume, deposits moisture at atmospheric temperature, and as the pressure is increased larger percentages of the contained water vapor are precipitated. Hence either high pressures or low temperatures increase the condensation of the water vapor.

In the exceedingly dry atmosphere of the Arizona and Nevada deserts, operating air compressors at a pressure of 100 pounds and cooling the compressed air only to atmospheric temperature always causes precipitation of water, and of lubricating oils from the cylinders, in the air receiver. That all of the moisture in the air is not deposited in the receiver is shown by the fact that in pumps driven by compressed air some moisture freezes in the cylinders and also forms frost on the exhaust outlet.

^a Burrell, G. A., Biddison, P. M., and Oberfell, G. G., Extraction of gasoline from natural gas by absorption methods: Bull. 120, Bureau of Mines, 1917, 71 pp.

The condensable fractions in natural gas, like those in air, are not visible under ordinary conditions, but at times the sudden release into the atmosphere of gas from confinement under pressure will cause vapors of hydrocarbons to form a mist resembling fog or steam.

In the treatment of natural gas for gasoline recovery the result desired could be accomplished either by compression or refrigeration, but the complex mixture of gases and vapors complicates the problem. Without an exact knowledge of the various members and the proportion of each to the whole, an attempt to calculate the most suitable pressures and temperatures becomes little better than a guess, and the most practical solution is by tests and experiments. According to the law of partial pressures,^a if gas contains 10 per cent of condensable vapor and is under a pressure of 200 pounds, the condensable 10 per cent has acting upon it a partial pressure of 20 pounds, or 10 per cent of the total pressure. As the proportion of different condensable vapors in the gas becomes smaller through their partial condensation, the partial pressure acting on them also becomes less. The percentage of the gage pressure acting on any one of the gas or vapor constituents varies in proportion to the percentage of volume occupied by that constituent. If the constituent is condensable, condensation will lower its proportion to the volume of uncondensed gas to a point at which the partial pressure acting on it is too small to cause further condensation, leaving a portion of the vapor uncondensed throughout the entire treatment. As the acting pressure becomes lower, condensation tends to cease, but lowering the temperature will cause further condensation. Under constant gage pressures and decreasing temperatures, the largest percentage of the heavier hydrocarbons contained in the gas is recovered, and the losses are confined to the lighter members.

Although all of the condensable vapor and even the true gases can be liquefied by increasing the pressure or reducing the temperature sufficiently, the application of these processes in treating natural gas for gasoline is limited by the commercial considerations. Extremes of either pressure or temperature would yield condensates too volatile for commercial use, also the machinery and other equipment required would be expensive to install and difficult to operate and maintain.

To recover the valuable hydrocarbon fractions that are held as vapors in natural gas, only such pressures and temperatures are necessary as can be obtained by the use of machines and fittings of standard construction and capacities. As the power used to compress the gas heats it, developing power by expanding the compressed gas cools it. By using the cooled gas as a refrigerant, the

^a Lusk, C. E., *Engineering thermodynamics*, 1912, p. 481.

temperature necessary, in conjunction with the pressures obtained, to extract the maximum of commercial condensate can be developed.

The temperature and the pressure that will yield the most profitable results are those that together will recover the largest possible amount of condensates of low vapor tensions and as much of the lighter parts as can be so blended or handled as to conform with legal standards of safety and make a good motor fuel

ABSORPTION THEORY.

The absorption process for recovering condensate from natural gas is based on an entirely different physical property, that of the solubility of vapors in liquids. In this process the gas is brought into intimate contact with a liquid which dissolves or absorbs the vapors; these vapors are later distilled from the absorbing medium and condensed.

USE OF COMBINATION COMPRESSION AND REFRIGERATION PROCESS.

As shown above, the fundamental principles of the compression process are compression and cooling of the natural gas to pressures and temperatures at which certain hydrocarbons condense.

Plants of this character are erected to treat casing-head gas from oil sands or from sands closely associated with oil, the gas being brought to the surface either between the casing and tubing of an oil well or with the oil in the tubing. The quantity of gas from each well is usually comparatively small and in some installations as many as 500 or 600 wells are connected with one compression plant of not more than the average capacity.

The dry (treated) gas is, at most plants, used on the oil leases to drive the gas engines of the compression plant, and also for gas and steam engines in pumping and drilling wells. A few compression plants sell the treated gas for commercial use in cities or for manufacturing purposes. The cost of pipe lines and equipment necessary to deliver the small quantities of gas to market would, in general, be excessive. There is seldom much gas left after the quantity necessary for furnishing power has been used.

The value of the gas for heating, power, and lighting is not impaired appreciably by removing the gasoline content. If this gas were not treated, the gasoline would, at most leases, be burned with the gas used for power purposes and practically be wasted as far as serving any useful purpose is concerned.

GAS LEASES.

Until 1914 practically all the gas being treated for its gasoline content was sold or leased under varying conditions to gasoline producing and marketing companies independent of the companies pro-

ducing the oil. Since that time the oil producing companies, realizing the profits to be made, have constructed compression plants to treat the gas produced with the oil on their leases, and in many instances are now purchasers or lessors of gas from surrounding properties.

Before 1914 gas leases were made mostly on a flat rate, which varied from 2 to 5 cents per 1,000 cubic feet. Often the contract stipulated the vacuum to be held on the oil wells, and specified that any treated gas not used to run the compression plant was to be returned to the lease from which it was taken, the return gas lines to be paid for and laid by the purchasing party. Other leases required that a certain percentage of the gas delivered to the compression plant should be returned, this figure in some instances being as high as 80 per cent.

SLIDING-SCALE LEASE.

One lease in Pennsylvania required that 7 cents per 1,000 cubic feet of gas be paid the lessor when the price received by the lessee for gasoline in tank car lots during the preceding month averaged 10 cents per gallon or less, and an advance of 1 cent per 1,000 cubic feet for each advance of 2 cents in the monthly average price of gasoline. The gasoline company had made arrangements to sell the treated gas at 12 cents per 1,000 feet to a company which supplies gas to near-by towns. This was one of the few plants found by the writer that made such disposal of the dry gas.

Such a form of contract, known as the sliding-scale lease, is being used almost entirely at present in the Mid-Continent fields, except that usually the contract stipulates that gas after treatment shall be returned to the lease from which it was originally taken. It also provides that all gas necessary for power to operate gas and water pumps as well as the compressor plant, shall be taken from the treated-gas lines at no cost to the lessor.

At the present time (December, 1916) leases in Oklahoma are being made on what is known as the 4-10 or the 5-10 basis, meaning that the price paid for the privilege of removing and selling the gasoline from the gas is to be 4 cents or 5 cents per 1,000 cubic feet when the price received for the product is 10 cents per gallon, with an increase of 1 cent per 1,000 feet of gas for each additional 2 cents per gallon, as in the Pennsylvania lease previously mentioned. On the 4-10 scale, with gasoline selling at 18 cents per gallon f. o. b. the plant, the price paid for gas would be 8 cents per 1,000 feet.

WEAK POINTS OF LEASE.

There seems to be two very weak points in such a form of contract; one is that no account is taken of the present or future quan-

tity of condensate in the gas, and the other is that as the wells and field grow older the percentage of gas returned to the original owner will become progressively less and eventually will become nil. In the Glenn pool, Oklahoma, this point has been reached at some plants.

In one plant, to the knowledge of the writer, the volume of treated gas discharged is at times insufficient to furnish fuel for the engines running the pumps and compressors. This condition is quite usual in older districts such as Sistersville, W. Va., where dry gas is bought to operate the compression plants, or, as at one plant, electric power is purchased and is used to drive the compressors.

As the quantity of gas decreases, the owner will receive less for the gas and have less treated gas returned to him, being forced eventually to buy gas for lease power while the plant operator will, for a long time at least, make the same or a slightly decreased quantity of marketable condensate. It would seem that these conditions should be taken into account in the gas contract.

ROYALTY LEASES.

Royalty leases are made and held in some fields, particularly in California, but do not seem to have been generally used throughout the United States.

Before the compression process and its results became well known and understood, gas could commonly be obtained on a royalty of one-eighth of the value of the product marketed, but as the gas producers learned of the treatment and profits, royalties have increased. In California, as much as one-third of the value of the product for gas containing less than 2 gallons of gasoline per 1,000 cubic feet is being paid. The writer is reliably informed of a contract in Oklahoma under which a royalty of one-half is being paid, the gas producing between 4 and 5 gallons of gasoline per 1,000 cubic feet.

FACTORS CONTROLLING ROYALTIES AND LEASES.

The royalties and prices that can be paid for gas under lease for gasoline recovery depend on the following conditions:

Wells scattered over a wide area will necessitate an expensive gathering system, including pipe lines and pumps or booster stations, and the cost of operation and upkeep. The quantity of condensate in the gas is vital because the cost of a plant and of plant operation is practically the same for rich or lean gas. If the gas is to be returned to the original owner for use on the lease, the cost, upkeep, and operation of the return or "dry gas" line are to be considered. The distance from a railroad station and the length of pipe lines required to bring the blending naphtha to the plant and transport the product to the station, also the distance from a market, and the source of naphtha supply will have a bearing on production and marketing costs.

The time the lease or contract is to run is also a factor which directly affects the price that is to be paid for gas, as the total cost of installation must be paid out of net receipts by the time the contract expires.

In general, all the factors entering into any manufacturing enterprise must be taken into consideration when the price to be paid for gas leases or royalties is being set.

METHODS OF COLLECTING NATURAL GAS.

The term "casing-head gas," when strictly applied, means gas coming from an oil sand between the casing and the tubing through which the oil is pumped. This term, however, as used in connection with compression plants, has been broadened so as to apply to any gas rising with oil in the tubing, or from flow lines, and to vapors coming from oil in traps or flow tanks.

In eastern practice it is usual to collect only the gas coming up between the casing and tubing in wells which are usually held under high (20 to 26 inches of mercury) vacuums, no attempt being made to collect the gas coming up with the oil in the tubing or the light fractions given off in flow or storage tanks. Oklahoma operators collect gas at the casing head, and in some instances from flow lines and flow tanks. In California an entirely different practice has developed, because of the soft running oil sand in most California fields. The oil and gas from one or more pumping or flowing wells is led into a specially built tank, called a trap. Traps of different designs are so constructed and operated as to permit the separation of the oil and the gas at pressures either above or below atmospheric, as circumstances may require.

USE OF TRAPS.

Originally traps were invented and designed to work under pressure with the object in view of separating, collecting, and saving the gas and also the oil atomized and mechanically carried with the gas into pipe lines or into the atmosphere; also to hold in the oil the gasoline fractions that would otherwise be volatilized when the pressure on the oil was released; and, further, to allow the oil to absorb as much of the condensable fractions from the gas as possible, thus raising the gravity of the oil and in a measure saving the most desirable fractions of the crude product. As an instance of the use of a trap and its effect on the oil produced, F. B. Tough, of the Bureau of Mines, cites the use of one at a gusher in the Midway, California, field. This well when flowing through a trap produced oil having a gravity of 32.5° B., and when flowing uncontrolled under considerable rock pressure into a collecting sump produced oil, as collected in the sump, with a gravity of 26° B., a gain of over 6° B. resulting from the use of the trap.

Traps are of interest to the casing-head industry principally as separators of oil and gas, or, as developed and used by at least one com-

pany, for holding a vacuum on the oil to separate from it such fractions as will distill off at the normal temperature and under a vacuum of 5 to 15 inches of mercury, thus reversing the usual effect and method of trap operation.

TRAPS HELD UNDER VACUUMS.

This company had found that the crude oil during transportation and storage and in transfer to the stills of the refinery lost a part of the lighter fractions; also, the temperature used in cooling the still vapors permitted further losses of the lighter fractions, which could be collected as vapor and saved by treating in a compression plant with the gas and vapors produced at the wells.

The method adopted is to relieve the crude oil of its lighter gasoline fractions by holding a vacuum on the trap, as oil is sprayed or flows into it with the casing-head and tubing gases, and treating all the gas and vapors thus obtained in the compression plant. The condensate, which has a gravity of 86° B., is held under a pressure of 10 to 15 pounds and kept cool by shading or insulating the storage tanks until the product is blended with naphtha.

This company has three different production conditions to

meet in the use of traps as follows: (1) Shallow pumping wells which have little or no gas pressure, (2) deep pumping wells with varying pressures of gas, and (3) deep flowing wells which have more or less pressure

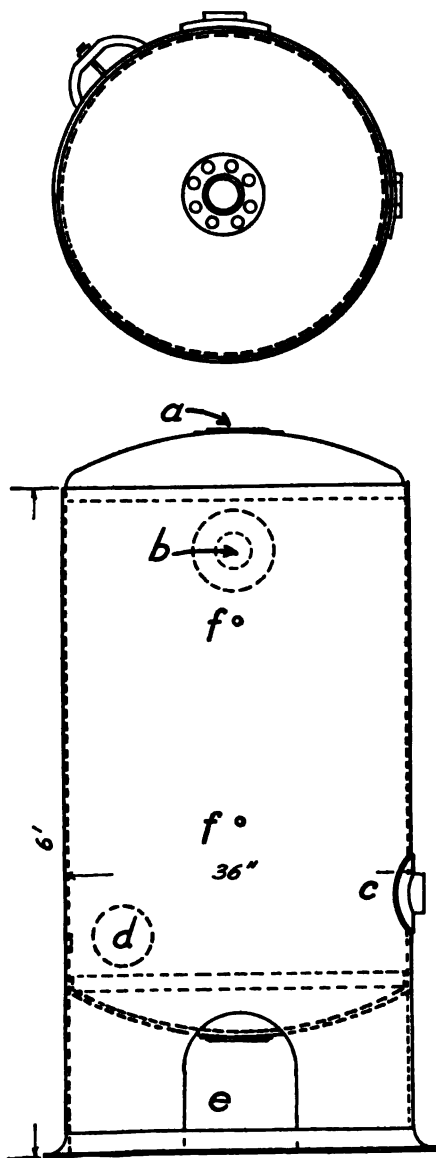


FIGURE 4.—Trap used with shallow pumping wells.
a, gas outlet; b, gas and oil inlet; c, oil outlet;
d, manhole; e, clean out; f, holes for gage glass.

on both gas and oil at all times and flow by head. Each condition is met by the use of traps of somewhat different design and operation.

Oil and gas from the shallow pumping wells is carried through flow lines, each controlled by a check valve, to a manifold, from which all the oil and gas goes through one line to a simple trap (fig. 4), the gas drawn out at the top at 5 to 15 inches vacuum to the scrubbers and vacuum pump, the oil being drawn off at a point near the trap bottom, to storage.

Production from deep wells flowing or pumping by "heads," which often builds up pressure in traps and lines by rushes of oil and gas too large for the normal capacity of simple traps and vacuum pumps, is handled by traps of special design, shown in figure 5. Two lines are laid from the well; one from the well tubing carries oil and gas pumped, or flowing together, the other line carries gas from between the casing and tubing. These lines are joined a few feet back of the trap intake, thus forcing all the gas and the oil to flow together into the trap at a point near the top; the gas separates from the oil and flows out at the top of the trap to the vacuum pump. The oil flows out near the trap bottom through a Crane balanced globe valve. The stem of the valve is rigidly connected to a counterbalanced float tank, which rises as the oil in the trap becomes low, closing the valve; fills and pulls the valve open when the flow increases and the oil rises in the trap. The float tank is connected both to the top and the bottom of the trap, as shown in figure 5, by swinging

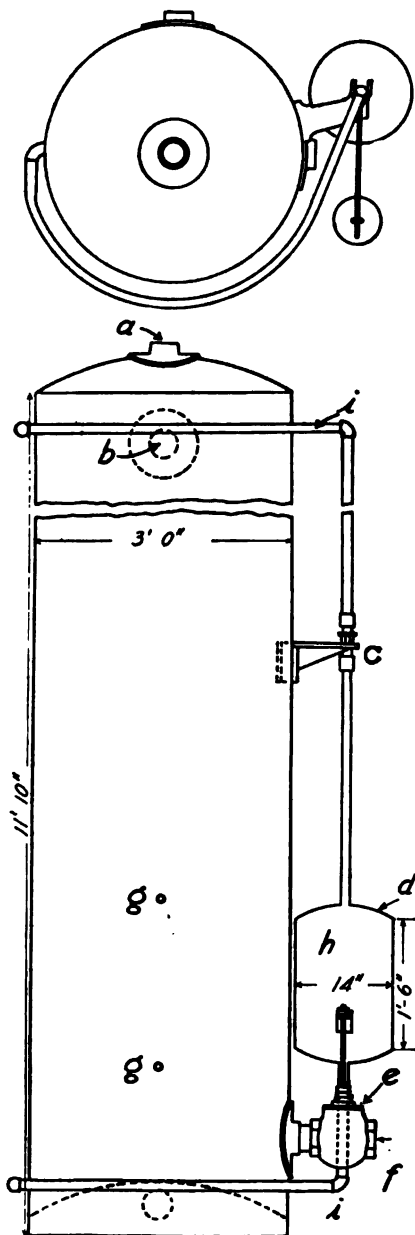


FIGURE 5.—Trap used on flow lines under pressure. *a*, gas outlet; *b*, gas and oil inlet; *c*, ball counter balance; *d*, float tank; *e*, 3-inch Crane balance globe valve; *f*, oil outlet; *g*, holes for gage glass; *h*, chamber; *i*, swinging pipe connections.

pipe connections. As the oil rises and falls in the trap oil flows in and out of the float tank through the bottom pipe connection. The top pipe connects the float tank and the top of the trap and acts only as a pressure equalizer. This trap is of standard design and is marketed by supply companies. The trap shown in figure 6 was

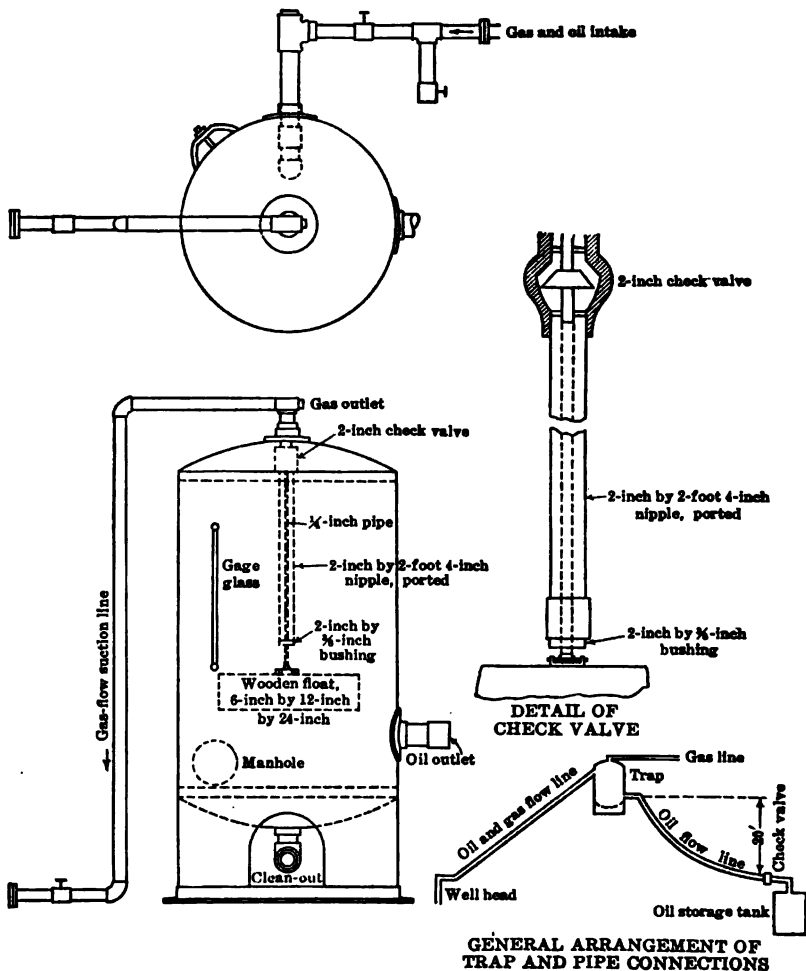


FIGURE 6.—Trap used with flowing wells.

designed by the company using this system of holding vacuums on traps. It is used, particularly with flowing wells, in much the same way as the trap shown in figure 5 and operates automatically.

When the oil level rises above a certain point the gas discharge shuts off entirely. This permits pressure to build up on both intake and discharge oil lines, causing a back pressure on the intake and a greater flow from the discharge; the trap thus clears itself and

returns to normal operation. It is set, as shown in figure 6, above both the well head and the storage tanks, so that when the oil and gas flow is small and a vacuum is built up in the trap the 20-foot drop and the horizontal check valve in the discharge line keep the oil from backing up into the trap, and permitting air to enter the trap and the gas lines.

As an example of loss of the light fractions of oil in storage, a 55,000-barrel storage tank filled with Cushing crude oil is reported by A. N. Kerr, of the Riverside Western Gasoline Co., to have lost by evaporation 9 inches, or 2.5 per cent of its total content, in one year. The loss, without doubt, consisted entirely of the light fractions that make up the gasoline content of the crude oil as it comes from the wells. Much of this loss might have been saved by using vacuum traps or by covering the tank and connecting it to the intake line of a compression plant.

FIELDS IN WHICH TRAPS ARE USED.

Traps are used more universally in the California fields than in other fields throughout the United States, because in California oil is sold on a gravity basis, making it desirable and profitable to retain as large a proportion of the light fractions as possible in the oil in order to keep the gravity at a maximum. The general practice, with the exception of the company using vacuum traps, previously mentioned, is to hold pressure on traps to save the light or gasoline fractions.

Plate I, *B*, shows a view of the Starke trap, designed and patented by Dr. Eric A. Starke, of the Standard Oil Co. of California, and used by that company.

Plate I, *C*, shows the McLaughlin compound trap, capacity 800 barrels of oil and 7,000,000 cubic feet of gas per day at 300 pounds pressure, patented by A. C. McLaughlin, of the Kern Trading & Oil Co., on the left, and the McLaughlin single-chamber trap, on the right. Plate III, *A* shows the cone chamber, McLaughlin trap.

Simple cylindrical traps used in the Fullerton field in conjunction with a casing-head gasoline plant are shown in Plate IV, *A*.

GATHERING LINES.

Gas from casing heads, tubing, traps, or flow tanks to be treated by compression is led through small (usually 2-inch) lines to a gas main that carries it directly to the plant or to a vacuum or pumping station.

At plants where the gas comes from near-by wells and the gathering lines are short, and it is not desirable to hold a high vacuum on the wells, the vacuum needed to carry the gas through the lines is

often developed in the low-pressure cylinder of the compression plant. By this method a vacuum pump is at many plants unnecessary, and the plant is simplified by requiring one less unit. From this condition the practice varies to plants at which gas is gathered from areas as large as 32 square miles, with 6 and 8 inch mains 7 to 10 miles long, and as many as 30 substations or gas pumps forcing gas through the pipe lines to the compression plant and holding the desired vacuum on the wells. In order to hold high vacuums on the wells from which gas is being drawn it is necessary to have the vacuum pump close to the wells, as it is impossible to maintain a minimum pressure through pipe lines of great length.

In designing a plant or plants to treat gas coming from a large area the number of plants, the number of vacuum and booster stations, the situation of the plant or plants, and the size and length of the pipe lines require careful mathematical calculations to determine the most economical installation. Many plants visited by the writer showed that little or no consideration had been given to these features. Such conditions may be due partly to the use of gas lines laid before the treatment of gas was considered and the reversal of the direction of flow in parts of such lines in order to bring the gas to the plant. The fall in pressure as the gas flows through the pipe lines and the increase in volume caused by the lowered pressure and the increased number of lead lines from wells as the gathering progresses toward the plant, make it essential to use pipes of progressively increasing diameter as the gas approaches the plant.

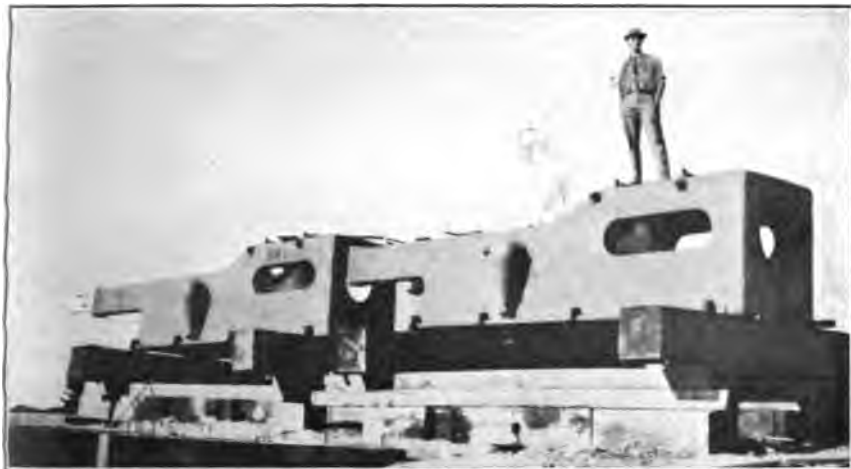
If vacuum pumps and booster stations are placed at points in the gathering lines to reduce the volume and increase the pressure of the gas, the lines leading from such plants may be smaller than those transmitting the gas under lower pressures.

In practice the sizes of gas mains have been determined usually by rule of thumb and bear little relation to the actual volumes, gravities, and pressures of the gas.

FLOW OF GAS IN PIPE LINES.

In designing gathering systems of compression plants to treat or pump gas, the determination of the proper size of pipe to use in lines for carrying a given quantity of gas at known pressures is so essential that the writer believes the formulas covering these points will be of value and interest to those engaged in compressing gas. Formulas and tables on the flow of gas in pipes have been developed by T. R. Weymouth, of Oil City, Pa., and discussed by him in a paper^a written for and published by the American Society of Mechanical Engineers. These formulas, with that part of the paper that treats of the flow of gas in pipe lines, are given on pages 107 to 112.

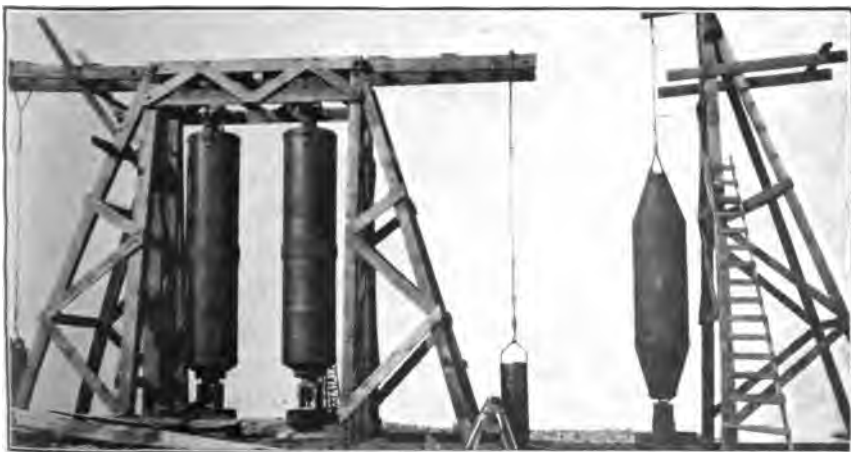
^a Weymouth, T. R., Problems in natural-gas engineering: Trans. Am. Soc. Mech. Eng., vol. 34, 1912, pp. 185-234.



A. TWO SINGLE ENGINE-BED CASTINGS, WEIGHT 31,000 POUNDS EACH.



B. STARKE TRAP AND CONNECTIONS, USED WITH GUSHERS MAKING LARGE QUANTITIES OF OIL AND GAS.



C. McLAUGHLIN COMPOUND TRAP (ON LEFT) AND McLAUGHLIN SINGLE-CHAMBER TRAP (ON RIGHT).



4. VIEW OF COMPRESSION PLANT.

Note tanks at left and cooling tower at right of compressor building.



2. COMPRESSION PLANT, DAILY CAPACITY 7,500,000 FEET, BUILT ON SLOPING GROUND, HAS GRAVITY SYSTEM FOR HANDLING WATER AND CONDENSATE.

DRIPS AND SCRUBBERS.

Particles of crude oil are often carried into gas lines from flow lines, flow tanks, and traps; these, with fractions of heavy vapors condensed by changes in temperature and pressure of the gas in the transmission lines would, if not removed, cause trouble in the pipes, damage machines and meters, and discolor the condensate produced in the first-stage accumulators.

DRIPS.

Drips are placed at the low points in gas pipe lines to collect and remove condensed moisture and naphtha vapors and crude oil carried into the line, which if permitted to accumulate would restrict the passage and might do considerable damage by being forced through the line as a slug, forming a powerful hammer. The type of drip usually found is constructed of one or two lengths of 6 or 8 inch pipe capped at both ends and connected by a 1 or 2 inch pipe and valve, placed at the lowest point of a downward curve in the gas main. The liquids settling to this point drain into the drip and are either blown out and wasted, or, if the quantity and quality warrant saving, are collected in tanks and, after being distilled or filtered, are used for blending or are sold.

Officials of a plant visited by the writer gave the following data on "line distillates" collected by tank wagons from the gas-line drips:

Data on distillate collected from gas-line drips.

	Volume collected, gallons.
August, 1915.....	7,000
November, 1915.....	14,625
January, 1916.....	19,142
March, 1916.....	11,549

Average gravity for an entire year, 55° B.

The figures represent the total quantity of condensate collected during the month named and show the direct effect of atmospheric temperature on the quantity precipitated. The product as collected was discolored, but as all the condensate produced was shipped to the refinery with the crude oil in pipe lines, the color of the line distillate was of no importance.

SCRUBBERS.

The term "scrubber," as used in the casing-head gasoline industry, is restricted in its meaning to tanks for settling liquids out of the gas or to tanks fitted with baffles or some form of filter for removing liquids or dust from the gas, and is not used, as in the coke-oven or the artificial gas industries, to designate equipment in which certain constituents are removed by chemical action or absorption in liquids.

The usual form of scrubber is a vertical tank (see Plate IV, B) varying from 3 by 6 feet to 6 by 20 feet in size, with a gas inlet in

the side and near the bottom, the end of the inlet pipe being turned downward or fitted with a baffle, so that the gas discharges toward the bottom. The gas rises through the tank slowly, thus permitting particles of liquid or dust to settle.

Scrubbers of the filter type are of the same size and form as the settling tank, but have perforated horizontal partitions on which are various filtering mediums such as moss, hay, or sponge. When the filtering medium becomes saturated or dirty it is removed and burned, if moss or hay, or is cleaned and replaced, if sponge is used. Scrubbers are always put in pipe lines ahead of compression machines, meters, and the pump. They also serve as intake receivers for the gas pump or the compressor, one tank, 3 by 6 feet in size, being used on the intake of each machine, or a tank of corresponding larger capacity for the entire plant intake.

VACUUM STATIONS.

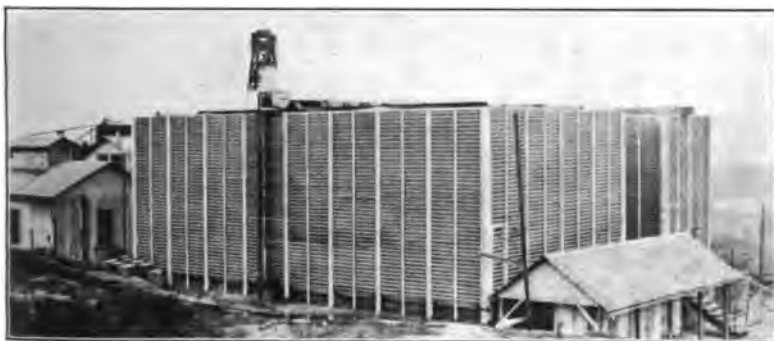
The number and the situation of vacuum pumps and booster stations on a gas-gathering system depend on the vacuum to be held on the wells, and the length and size of the pipe lines through which the gas is to be forced. When it is desired to hold the vacuum on the wells at the maximum, as is usual in the eastern fields, the vacuum pump must be placed near the wells, because friction, and often pipeline leakage, reduces the suction on the well. The most common vacuum-pump installation consists of a gas engine of 35 to 70 horsepower belted to a duplex pump having cylinders varying between 10 by 17 inches and 20 by 20 inches. Intake pressures vary between 14 and 26 inch vacuums, and discharge pressures between a vacuum of 5 inches and a gage pressure of 5 pounds.

The "booster," if used, is to be placed in the same building as the vacuum pump. Power is developed by a 25 to 70 horsepower gas engine, either belted or direct connected to a compressor used to increase the pressure and force the gas through field lines either to another pump and booster or to the main compression plant. Compressors used in this way take gas directly from the discharge of the vacuum pump, usually holding a light (zero to 5-inch) vacuum at the intake, thus relieving the pump from working against high discharge pressures. Gas pumps are built too light to pump gas against pressures exceeding 10 pounds and usually discharge at nearly zero gage pressure. Compressors used as boosters deliver gas to the lines at pressures of 20 to 40 pounds and at any temperature incident to the compression.

Keeping gas under positive pressure through as much of the gathering system as possible tends to prevent air being drawn in and contaminating the gas, although any pressure high enough to keep air from leaking in also permits some gas, charged with condensable



A. McLAUGHLIN CONE CHAMBER TRAP.

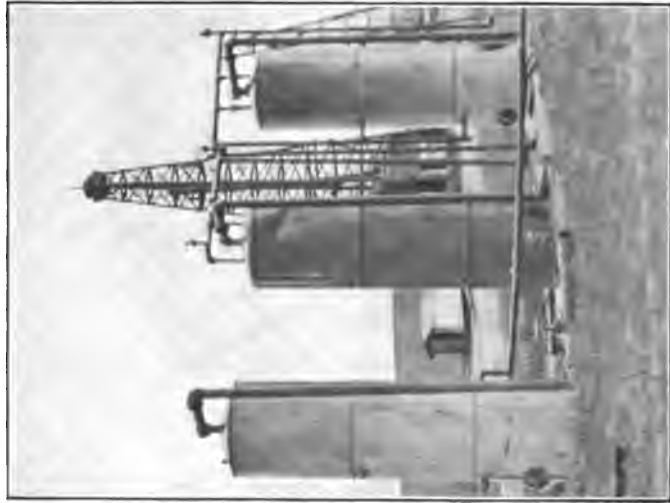


B. COOLING TANKS SET TO TAKE ADVANTAGE OF SLOPING GROUND AND OPEN-AIR CIRCULATION.

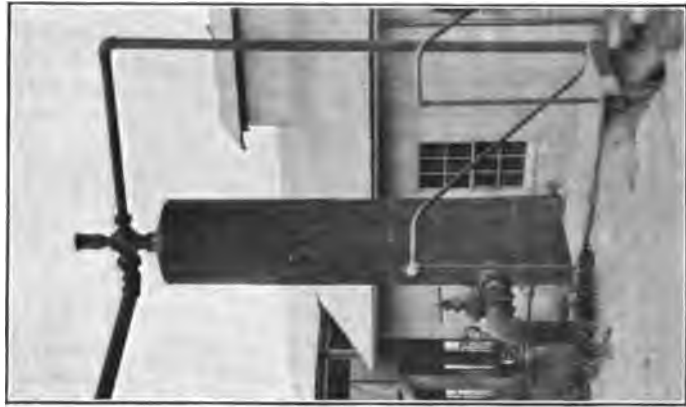


C. HIGH-PRESSURE AND LOW-PRESSURE ACCUMULATOR TANKS.

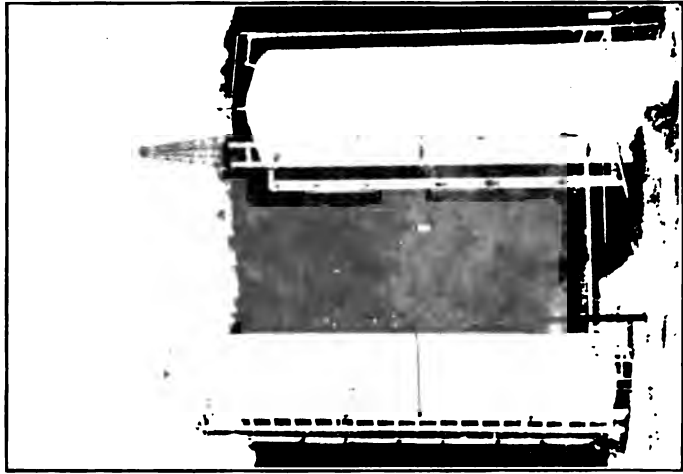
Note heavy double-riveted construction of high-pressure tanks.



A. SIMPLE CYLINDRICAL TRAPS.



B. INTAKE RECEIVER AT 1,000,000-FOOT PLANT.



C. "MAKE" AND STORAGE TANKS, SHOWING
PIPING AND PRESSURE VALVES.

vapors, to escape. Air in gas cuts down the volume of gas treated, necessitates the use of higher pressures to precipitate the condensate, and carries varying proportions of water vapor which condense with the gasoline. Another advantage of transmitting gas under pressure is that pipe-line capacities are increased, making possible the use of pipe of smaller diameter in the gathering system.

There is a decided tendency at the present time in favor of welded joints in pipe lines to carry gas either at vacuums or positive pressures. It is claimed that lines with such joints cost less in upkeep and repairs and can be maintained practically gas tight.

SITUATION OF PLANT.

Factors entering into the problem of the best situation for a compression plant to treat gas from a given area are so numerous and varied that no rules which would apply even in a general way to all conditions found in oil fields can be given.

The most important factors and those which will have the greatest weight in determining the most practicable situation for a plant are: Water supply, transportation facilities, and the length of gathering lines and return dry-gas lines necessary. In eastern fields an ample water supply can usually be easily obtained from a creek, river, or well, and causes little concern to plant operators, but in the Mid-Continent and the western gas-producing areas the question of water supply often becomes of major importance.

In the California fields some plants pay as high as 3½ cents per barrel for water, which has to be treated or condensed before it is fit for use in boilers or jackets. Other plants pump their water for distances of several miles, and at these the cost of installation, operation, and upkeep of the pumps and the pipe lines make the cost of water an important item. One California plant is using water separated from the oil in a dehydration plant on the cooling coils, and is distilling and condensing water for jacket use, all other sources of water having practically failed.

It is obvious that the nearer the plant is to the center of gas production the smaller will be the size of the pipe lines and the amount of power required to conduct the gas to the plant. However, other factors, such as water-supply and transportation facilities may affect the choice of a site. The plant should be so situated as to suit best all conditions, each being considered in regard to the others. Compression plants are often built on a hillside, advantage being taken of the natural slope to install a gravity system of handling the water and the condensate. The water, being drained from all points of use to one pond or basin, requires only one pumping unit to return it to the top of the towers or the storage tank above the plant, from which it is drawn for all purposes. Plates II, B, and III, B,

show plants using a gravity system for handling water and condensate. In this system the condensate can be run by gravity from accumulators and "make" tanks to storage and blending tanks. The accumulators do not have to be set in pits or cellars; this is a decided advantage, because gasoline vapor may collect in such pits, unless they are thoroughly ventilated, in sufficient quantity to form an explosive or asphyxiating atmosphere and endanger the safety of the plant and the men. At plants visited by the writer three men have lost their lives by suffocation in pits filled with gasoline fumes. Some plants have a rule that a man before going into a cellar containing accumulator tanks shall notify another workman, or, if there is known danger from leaks, men shall work only in pairs or within calling distance of each other. Plants are often built on a hill or rise because of the better ventilation and the cooling effect on water towers and coils.

DATA ON PLANT PRACTICE.

A number of tables compiled from data taken at each of the plants listed in Table 2 or given to the writer by owners or by officials of the various companies are presented herein.

In Table 2, column 1 shows the field in which each of the plants is situated. Each plant is given a number, shown in column 2, which it retains throughout all tables and discussions following. Capacity and production data of these plants are shown in columns 3 to 5. Column 3 shows the quantity of gas treated, in thousands of cubic feet, as estimated by plant operators, or computed from meter readings, or from the compressor displacement and the number of revolutions, or the pipe-line capacities at measured pressures. Column 4 represents the production, after weathering, of unblended gasoline, and is approximately the quantity of condensate sent to market. Tank-car outage (or loss in transit) has not been deducted, as that factor varies widely with the distance traveled by the cars and the temperatures encountered during the shipment. Column 5 shows the number of gallons of condensate produced from each thousand cubic feet of gas treated, based on the plant capacity and the total product sold, as indicated in column 4. The gravity in degrees Baumé, as given in column 6, is determined from the plant product as a whole, before blending, or computed from the gravity of the blend, the quantity and the gravity of the blending stock used being known.

TABLE 2.—Data showing situation, capacity, and output of the plants visited, and the gravity of the product.

Field where plant is situated.	Plant No.	Capacity (in 1,000 cubic feet).	Daily production, gallons.	Gallons produced per 1,000 cubic feet of gas.	Gravity, °B.
California fields:					
Fullerton	1	1,000	600	0.60	68
Do	2	350	1,200	3.40	76
Do	3	450	700	1.50	78-80
Do	4	1,250	1,600	1.30	72-74
Do	5	5,000	1,200	.24	
Santa Maria	6	2,500	5,500	2.20	81
Do	7	7,500	7,500	1.00	70
Do	8	500	500-600	1.00	70
Do	9	1,000	2,500	2.50	80
Do	10	750	1,200	1.60	74
Do	11	1,500	3,000	2.00	79
Do	12	700	1,500	2.10	81
Do	13	(a)			
Ventura	14	1,000	2,000	2.00	86
Do	15		800		
Salt Lake	16	250	325	1.30	65
Do	17	750	810	1.10	60-65
Do	18		1,500		68-70
Midway	b 13	1,800	1,400	.78	68
Do	20	1,500	1,400	.90	67
Do	b 21	1,000	700	.70	80
Eastern fields:					
Bradford, Pa.	22	600	1,200	2.00	96
Sistersville, W. Va.	23	200	800	4.00	84
Do	24	375	1,500	4.00	90
Do	c 25				
Do	26	500			88
Southern Illinois	27	750	500	.75	83
Do	28		200		83
Do	29				
Mid-Continent fields:					
Glenn pool	30	750	3,000	d 4.00	
Do	31	375	1,500	d 4.00	77
Do	32	3,000	22,500	7.40	84
Do	33	1,800	9,000	5.00	82
Do	34	750	5,250	7.00	82
Do	35	350	1,590	4.52	96
Do	36	2,000	2,310	1.10	78
Do	37	400	2,565	6.40	85
Do	38	750	5,055	6.70	
Do	39	200	510	2.50	
Do	40	250	600	2.40	
Do	41	350	550	1.60	
South Glenn pool	42	450	1,200	2.40	
Morris	43	300	300	1.00	
Do	44	500	1,850	3.60	
Beggs	45	350	710	2.10	
Muskogee	46	300	730	2.44	
Do	47	350	300	.85	
Glenn pool	48	1,200	2,380	2.00	
Cushing pool	49	2,000	2,810	1.40	
Cleveland	50	1,200	2,000	1.67	
Muskogee	51	250	925	3.70	
Do	52	250	500	e 2.00	
Cushing	53				
Nowata	54	f 2,250	10,000	4.00	
Do	55	1,250	1,600	1.30	
Do	56	380	600	1.60	
Do	57	830	1,500	1.60	
Morris	58	2,000	3,000	1.50	
Do	59	400	725	1.90	
Nowata	60	1,000	1,900	1.90	
Do	61	500	1,100	2.20	
Do	62	280	420	1.50	
Glenn pool	63	487	1,948	g 4.00	
Do	64	515	2,062	g 4.00	

a Plant not in commission.

b Plant capacities doubled and expansion sets installed or improved; product per 1,000 feet of gas increased.

c At this plant gas is compressed in one stage to a pressure of 150 pounds.

d Estimated.

e Eight per cent water with high and low pressure product.

f Thirty per cent air.

g Volume of gas estimated from product at 4 gallons per 1,000 cubic feet, a conservative estimate for gas from wells in the Glenn pool.

TABLE 2.—*Intake showing situation, capacity, and output of the plants visited, and the gravity of the product—Continued.*

Field where plant is situated.	Plant No.	Capacity (in 1,000 cubic feet).	Daily production, gallons.	Gallons produced per 1,000 cubic feet of gas.	Gravity, °B.
Mid-Continent fields—Continued.					
Glenn pool.....	65	90	350	a 4.00	-----
Do.....	66	90	351	a 4.00	-----
Do.....	67	98	384	a 4.00	-----
Do.....	68	136	545	a 4.00	-----
Do.....	69	126	500	a 4.00	-----
Do.....	70	141	568	a 4.00	-----
Do.....	71	712	2,884	a 4.00	-----
Do.....	72	375	1,500	a 4.00	-----
Do.....	73	466	1,884	a 4.00	-----
Do.....	74	124	498	a 4.00	-----
Do.....	75	173	690	a 4.00	-----
Caddo (Louisiana).....	76	2,000	3,600	1.80	79
Do.....	77	400	640	1.60	79
De Soto (Louisiana).....	78	45	350	7.00	73
Caddo (Louisiana).....	79	250	1,050	4.20	b 80
New Jersey: Refinery.....	80	1,750	5,400	3.09	76-83

a Volume of gas estimated from product at 4 gallons per 1,000 cubic feet, a conservative estimate for gas from wells in the Glenn pool.

b 50 per cent of this product was from low-pressure cylinder and 50 per cent from high-pressure cylinder and expansion coil.

In the following pages the writer has endeavored to describe as nearly as possible the treatment of natural gas in the recovery of gasoline by the compression process, and the mechanical units used.

The controlling functions of low and high pressure compression units of the plants studied by the writer are shown in Table 3.

Plant 2 is described in detail in pages 63 to 65. Plant 5 is a gas-pumping station, the condensate collected in cooling the gas before transmission is noted as production in Table 2 (p. 29). This gas after transmission through the pipe lines is treated in an absorption plant. Plant 10 uses three-stage compression. Plant 25 uses one-stage compression, compressing to a pressure of 150 pounds. Plant 80 is a refinery at which uncondensed still vapors are treated by compression; this plant is described in detail in later paragraphs.

PLANT INTAKE.

Table 3 shows that the temperature and the pressure at the plant intakes vary widely. The pressures tabulated as pressure at plant intake represent the pressure shown at the suction of the first-stage compression cylinder. Vacuum pumps, even if installed at the plant, are considered as part of the gathering system, and vacuum at plants where the compression cylinders are used to hold vacuum on the gathering system is taken as part of the compression operation and not as a part of the gathering system to which it rightly belongs. The intake pressures used vary from 18-inch vacuum to 5-pound pressure, and the temperatures from 60° to 125° F., the average

pressure being approximately that of the atmosphere, and the average temperature 80° F. The temperature of 60° F. noted above was recorded at plant 1, where the gas is cooled in coils placed in a water tower before it is compressed, and the temperature of 125° F. at a plant where a vacuum-pump discharge is used as the first-stage suction without cooling.

PRECOOLING.

Cooling the gas before it enters the first stage of compression, although not generally practiced, has distinct advantages. Reducing the temperature of the gas decreases the volume, the intake pressure remaining the same, so that more gas is taken into the compressor cylinder at each stroke, thus increasing the efficiency.

In warm climates plants having gathering lines, traps, and scrubbers exposed to the sun, which may heat the incoming gas to temperatures higher than 110° F., or having gas pumps and "booster" compressors in the gathering systems with no cooling coils after such units and before the plant intake, can by installing coils or other cooling apparatus increase both the plant efficiency and the carrying capacity of the pipe lines. Cooling the intake gas 30° or 60° F. will increase the quantity treated by a compression unit 5 to 10 per cent and lower proportionally the temperature of the gas discharged from the compressor. (See Table 11, p. 112.)

A plant in California lowered the temperature of the gas about 40° F. by passing it through a water-cooled coil ahead of the low-pressure intake. On the day of the writer's visit 1,250,000 cubic feet of gas was handled and the temperature lowered from 114° to 74° F. The coil used consisted of twelve 3-inch pipes 25 feet long, in 10-inch headers, placed horizontally in the cooling tower below the high and the low pressure gas coils.

A plant in Louisiana uses water cooling ahead of the vacuum pump and the first-stage compression intakes. In gas transmission gas piped to market is always cooled before being permitted to enter the mains, in order to increase the capacity of the line and to precipitate as much condensate and water vapor as possible.

38.	150	5		45		315	2.10	.420		250		150	380	2.56		
39.	100	0		45		790	4.85	.650		250		160	1,470	9.20	1.23	
40.		6		45		147	2.70	.690		250		55	280	5.20	1.14	
41.	55	9		25		210	4.20	.840		250		50	610	12.4	2.44	
42.	80			45						250						
43.		5		45						250						
44.		8		40		610	1.35	.270		275		450	975	2.16	.43	
45.	150			48		505	3.36	.404		300		150	505	3.36	1.404	
46.	105	4-8		40		505	4.80	1.36		300		105	505	4.80	1.36	
47.	175	4-8		40		540	4.80	1.01		250		175	840	4.80	1.01	
48.	300	4-8		80						()						
49.	90		70-76	80		1,760	5.9	.38		()						
50.	90		70	90		590	11.80	1.47	65	()						
51.	90		80	75		590	11.80	13.00	80	()						
52.	52	0		50		125	2.42	.50	75	()						
53.				43		253		.189	80							
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f Varied from positive pressure of 5 pounds to vacuum of 5 inches.

g Pounds, positive pressure.

h No coils used between low-stage and high-stage compression.

i At coil intake.

j Single-stage plant, no blend up.

a Ammonia process used from this point on.

b Atmospheric.

c Low-pressure cylinder: three stages used.

d Intermediate cylinder.

e 0.920, low compression; 0.430, intermediate compression.

FIRST-STAGE PRESSURES AND TEMPERATURES USED.

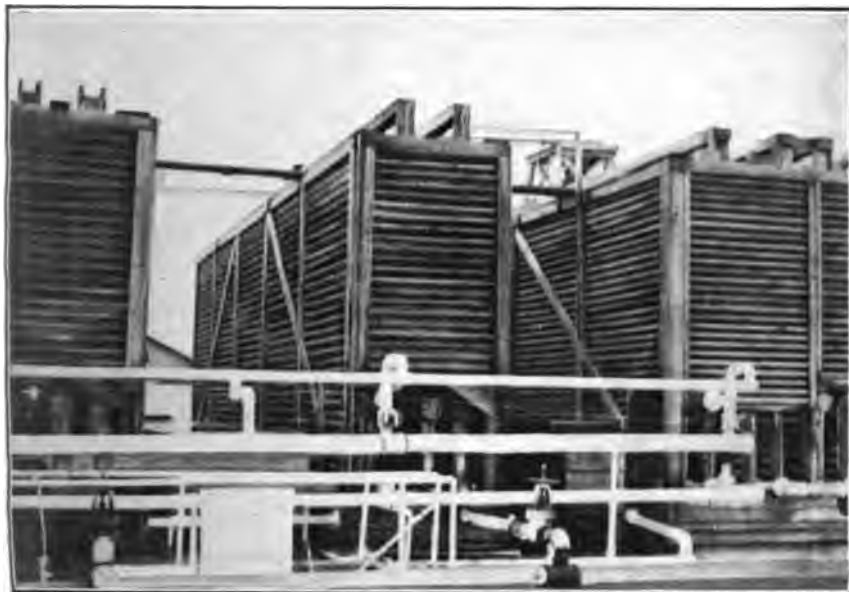
In plants using two-stage compression, the average pressure developed by the first stage is between 40 and 50 pounds per square inch; the temperature rises to between 200° and 250° F., depending on the temperature of the gas at the compressor intake and the number of compressions developed.

The increase in temperature of the gas from compression is a function of the power used to raise the pressure to the desired point. The power used depends on the number of compressions through which the gas is forced between its initial and final volume. As the amount of power actually expended, and not the initial and the final pressure, determines the rise in temperature the temperature increase due to a given number of compressions should be the same. If the intake temperatures of the high and the low stage compressor cylinders are equal, the final, or discharge, temperature of both cylinders of a two-stage compressor or of two single-stage machines acting as a high and a low stage unit should be the same, provided they are of equal horsepower and working properly and under uniform conditions. This is found to be approximately true in plant practice, as is shown by the temperatures of the compressor discharges given in Table 3.

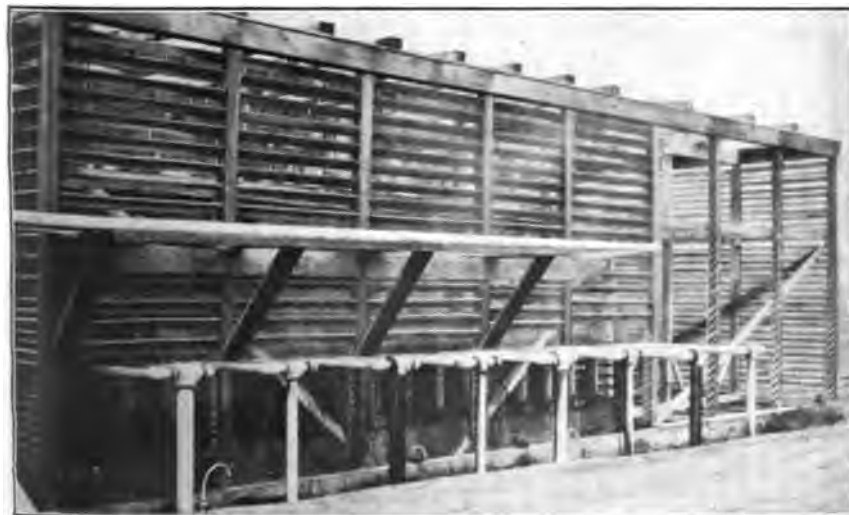
The line carrying the compressed gas from either the high or the low stage cylinder is usually fitted with safety valves set to pop at pressures of 5 to 20 pounds above the desired pressure in order to protect the machine in case a valve farther along in the system may have been left closed or a pipe in the coils or other units in the gas circuit should become stopped or clogged, thus causing pressure to build up throughout the system. In plants using expansion engines this condition may occur at any time through the engine valves freezing, thus stopping the usual discharge.

ATMOSPHERIC COOLING.

The compressed gas discharged from the compressor is carried through an oil separator, placed just ahead of the cooling coils, which traps any lubricating oil vaporized in the cylinders or carried mechanically with the gas, the oil being condensed by the time the gas reaches the trap. Cooling to this point is effected by radiation to the atmosphere surrounding the gas discharge pipe and to the jacket water around the compressor cylinder. Some operators are employing air cooling extensively in order to save water, the practice being to use a much larger pipe, exposed as far as possible to the atmosphere, for conveying the hot gas from the compressor discharge to the water coils, thereby reducing the speed of flow and so increasing the time of transmission that the temperature of the gas is materially lowered. One plant with a daily capacity of 1,250,000 feet and using 6-inch pipe to carry both high and low



A. COOLING TOWERS, MANIFOLD INTAKE, AND DISCHARGE LINES AND HEADERS.



B. HIGH AND LOW PRESSURE COILS WITH INTAKES ALTERNATING.

pressure gas to the coils, reduces the temperature between the compressor discharge and the coil intake 80° (240° to 160° F.) even in warm weather. In cold weather the reduction is greater.

Methods of air cooling are being rapidly developed and installed in districts where water is scarce and expensive, or the supply uncertain.

COOLING COILS.

In the eastern fields the cooling coils are generally of the continuous, submerged type, consisting of a 2-inch pipe 20 feet long, submerged in a tank of water. The total length of the coil varies between 300 and 500 feet, and the radiating area is 0.6 to 0.7 square feet per 1,000 cubic feet of plant capacity, and 2 to 5 square feet per horsepower used in compression. A continuous flow of water through the tank cools the gas to approximately the same temperature as the water.

In western and Mid-Continent practice a different method has been developed. Because of the difficulty of obtaining a continuous supply of cool water, it is necessary to use the same water over and over again in a closed circuit, the water being cooled in towers or in sprays over the principal storage pond or basin.

COOLING WATER BY EVAPORATION AND RADIATION.

Water exposed to air cools in two ways—by radiation, as long as the water is warmer than the air, and by evaporation.^a To obtain the greatest cooling effect the water must be so exposed to the air as to present the greatest possible surface. At some plants the water is cooled in towers (Pl. V, A) by means of sprays, or by permitting the water to fall in small streams on wire netting or screens, usually placed above the coils, thus atomizing the water and presenting a large surface to the air. The falling spray is often collected by V-shaped troughs that are placed a few inches above the top pipe of the coil and direct the flow of cooled water over the entire series of pipes. The water, dripping from the lowest pipe of the coil, is collected in a shallow basin beneath the coil and the tower, pumped to the top of the tower, and used again.

Plates VI and VII show views of a compression plant and the general arrangement of towers, compressor building, and storage tanks.

Some plants use a spray over a pond. The water is collected beneath the coils and conducted to a cistern from which it is pumped through upward sprayers placed over the storage pond, thence it is again pumped over the coils. The finely divided particles of water in the spray are cooled by radiation and evaporation while falling into the main body of water below. This system, although producing

^a Hausbrand, E., *Evaporating, condensing, and cooling apparatus*, 1903, 400 pp.

satisfactory temperatures, wastes more water than the tower installations, more being carried away mechanically by the wind; there is also a waste due to seepage, if the pond is of earth. The surface of the water in the pond is often exposed to the heat of the sun, which warms the body of water after cooling and before use over the coils. The sun has a decided effect in raising the temperature of the water in regions where the weather is warm during a large part of the year, as in Oklahoma and southern California.

WATER COOLING IN TOWERS.

In towers, during hot dry weather, temperatures 10° to 40° F. below that of the atmosphere are at times obtained with minimum losses of water. Many plant operators who have not experimented to find the amount of water that gives the lowest temperature to the gas being treated, use far more than is needed or gives the best results. Water falling in excessive quantities in a tower does not acquire the lowest possible temperature from radiation or evaporation, and the large bulk of water flowing over the coils can not cool the gas as efficiently as a smaller amount does, for the same reason. The best results are obtained when minimum quantities of water are circulated and finely divided while passing downward through the tower. Only enough of the cooled water should be directed onto the pipe coils to keep all parts thoroughly wet, thus giving the water the greatest possible opportunity to evaporate.

The use of auxiliary cooling agents such as ammonia or ammonia and brine is discussed later in connection with the descriptions of plants using such methods.

It may be of interest and use to operators to note here the latent heats of vapors being treated in the cooling coils. Burrell^a gives the following values for the latent heats of some petroleum distillates:

Specific gravities and latent heats of four distillates.

	Specific gravity, °B.	Latent heat, B. t. u. per pound.
Kerosene.....	43	105
Naphtha.....	56	103.5
Gasoline.....	65	100.6
Gasoline.....	89	100.2

In gasoline computations it is customary to use as the latent heat 100 B. t. u. per pound. Kent gives the specific heat of liquid gasoline of specific gravity 0.68 to 0.70 as 0.53 to 0.55, and Lucke^b quotes Regnault as stating that methane gas has a specific heat of 0.5929 at constant pressure and 0.4505 at constant volume.

^a Burrell, G. A., Personal communication.

^b Lucke, C. E., *Engineering thermodynamics*, 1912, p. 578.



VIEW OF 2,500,000-FOOT PLANT SHOWING ARRANGEMENT OF COMPRESSOR BUILDINGS, TOWERS, AND STORAGE TANKS.



ANOTHER VIEW OF PLANT SHOWN IN PLATE VI.

TYPES OF COILS AND CONNECTIONS USED.

Two systems of pipe connections are used between compressor discharges and cooling coils. In one system all low-stage compressors, and likewise all high-stage compressors, discharge the compressed gas into a common pipe or manifold, from which it is distributed by a manifold to the different sets of coils being used to cool the hot gas. (See figure 9, page 50.) The other system is to lead the gas discharged from each cylinder through a separate coil which cools only the gas from that compressor, the gas treated by each unit being kept separate throughout its entire circuit. The first method has the advantage of permitting one coil to be shut down or cut out for repairs while the compressor is running, the discharge from that machine being cooled in the coils still in use, or conversely, the coil may remain in use while the compressor is idle, and treat gas from other units. In the second method of connection if any coil or machine is out of commission the entire unit of which it is a part must be stopped, causing more of the plant to be idle.

Throughout the districts making natural-gas gasoline the 2-inch pipe in cooling coils is standard, and only a few plants using other sizes were found by the writer.

Systems and methods of connection vary widely, ranging from continuous return-bend coils in which all of the gas passes through the entire coil, to coils which divide first into two or more headers, and then into numerous separate pipes. In the multiple-header system each portion of gas passes through short lengths (20 to 80 feet) of the coil, and is again collected by headers before going to the accumulator tank and to the next stage of treatment. In the continuous system the gas passes rapidly through the total length of pipe, its temperature being reduced during the whole time of travel from end to end of the coil. In the multiple method the rate of flow in each pipe is reduced in proportion to the number of pipes used, so that the gas is sufficiently cooled while traveling through such a short length of coil.

There is much difference of opinion among operators as to which method gives the more satisfactory results. Many of the newest plants combine the two methods, as follows: Each coil unit is divided at the intake by a header into 3 to 12 sets of 2-inch continuous return-bend coils, each being 4 to 16 pipes high, and generally 20 feet long, depending on the cooling area desired. The intake header is placed horizontally at the top of the coil so as to allow the gas to travel downward in the same direction as the condensate. At the lower end of each set of return-bend coils the gas and condensate are again collected in a header before passing to the accumulator tank. Coils of this type and of the single-pipe continuous type have

the great disadvantage of being "hard to get at" in case any length of pipe in the coil needs replacement, whereas in multiple-header coils in which the ends of each pipe are packed in a gland, any member can be removed or replaced without taking down any other part of the coil.

DIRECTION OF FLOW OF GAS AND CONDENSATE.

It is universally conceded that the direction of flow of gas and condensate should be parallel and not countercurrent. As the liquid will drain toward the lowest point in the coil, the gas must enter at the top in order to flow with the condensate. As an illustration of the effect of counter flow, a plant in the Caddo field using a countercurrent flow of gas and condensate in the water-cooled coils produces no condensate whatever in the accumulator tank at the end of these coils. The gas yields 11.8 gallons of condensate per 1,000 cubic feet treated, but this condensate is all precipitated in double-pipe condensers cooled by expanded gas to a temperature of 38° F., in which the flow of gas and liquid are parallel. Undoubtedly some condensate would be precipitated in the water-cooled coil if the direction of gas flow was reversed. Other plants have obtained similar results under like conditions; one example being a California plant, in which the coil discharge pipe sloped upward to the accumulator tank, permitting condensate to gather in the bottom coils and discharge pipe and be constantly in contact with flowing gas. The accumulator tank was lowered so as to allow a drop in grade from the coils, with the result that the yield of condensate at that point increased noticeably. It seems that the condensate on long and intimate contact with the gas as above described, is again taken up by the gas as a vapor, even at high pressures, and carried to some point more conducive to precipitation and separation.

That condensate vaporizes in accumulator tanks has also been proved and has given rise to the practice of trapping off the liquid as soon as collected. Because of the facts stated, it appears to be the best and most productive practice to separate the gas and condensate as soon as possible after all the condensable fractions have been precipitated and always to allow the gas and condensate to flow in the same direction. If the above arguments hold true under all conditions, it would seem to be advisable to divide the gas to be cooled into a number of coils or pipes that will retain it only long enough to obtain the maximum cooling effect from the water used, and to separate the gas and condensate as soon as possible.

In order to divide the gas from a header equally in each of the coils or pipes of a coil, an orifice disk is sometimes placed in the intake from the header; the constriction causes a slight back pressure, forcing the gas to enter each coil in approximately equal quantities. The

size of the orifice is arbitrarily determined by making the sum of the areas of the orifice openings equal to the cross-sectional area of the pipe leading the gas to the header from the compressor discharge or the discharge manifold, as the case may be.

RADIATING AREA OF LOW-PRESSURE COILS.

In Table 3 the total surface areas of coils used in cooling gas from low-stage cylinders of various plants visited by the writer are tabulated, with the area per 1,000 cubic feet of gas treated per day; also, the area per horsepower used in compression. The average area of coil is between 0.6 and 0.7 square feet per 1,000 cubic feet of gas treated daily and nearly 3.5 square feet per horsepower used in compression. The latter factor is more useful for plant design, as the heat developed in compressing gas is a function of the power used in compression and not of the volume of gas being treated. A cooling area of 4 square feet per horsepower is usually used at gas-pumping plants and has proved satisfactory in most fields.

Column 10, Table 3, gives the temperature of the gas leaving the low-stage water-cooled coils at plants where such data was recorded and available. Minimum temperature should be maintained to precipitate the maximum percentage of condensate and benefit high-stage compression, as explained in previous paragraphs.

From these coils the gas passes to the low-stage accumulator tanks. Each coil may be provided with a separate tank or all the coils may be manifolded to one tank. Plate III, *C* (p. 26), shows the alternate arrangement of accumulator tanks of high and low pressure coils, each coil having a separate tank. The tanks are usually 3 to 4 feet in diameter by 6 to 10 feet high. The gas is led in at the side of the tank and near the top through a pipe turned or baffled downward inside of the tank, discharging at a point about midway between the top and bottom. The condensate settles to the bottom; the gas discharges at the top to the intake line of the high-pressure cylinder if each unit is independent or to the high-pressure intake manifold if that system is used.

High and low pressure coils, with intakes alternating, used at one plant are shown in Plate V, *B* (p. 34).

PRODUCTION FROM LOW-PRESSURE TREATMENT.

The proportion of condensate collected in the low-stage accumulator tanks averages 15 to 30 per cent of the total yield and varies between nothing and 40 per cent, depending on the content of condensable fractions in the gas and on the temperature and the pressure used. Some operators permit condensate to accumulate in the tanks until the entering gas is forced to pass through it, or to accumulate for a given time, and then run it into the storage tank through a hand valve.

The general practice, however, based on the theory that to separate gas and condensate as soon as possible produces the best results, is to remove the condensate from the accumulator tanks continuously with a small automatic trap that dumps often and with but little agitation keeps the tank practically empty at all times.

HIGH-PRESSURE TREATMENT.

From the low-pressure accumulator tanks the gas passes into the high-pressure cylinder of the compressor, and the cycle is repeated except that the higher pressure causes the lighter hydrocarbons to condense.

RADIATING AREA OF HIGH-PRESSURE COILS.

Table 3, which gives the average surface areas of both high and low pressure cooling coils at the plants visited, shows that a somewhat larger cooling area is used for the high-pressure gas. The area of high-pressure coils per 1,000 cubic feet of gas cooled daily was between 0.7 and 0.8 square feet and about 4.5 square feet per horsepower used in compression, whereas that of the low-pressure coils averaged between 0.6 and 0.7 square feet per 1,000 feet of gas and 3.5 square feet per horsepower. If the same amount of power is used by each cylinder of the compressor, there seems to be no reason why one cooling area should be larger than the other, unless the gas was not cooled in the low-stage coils to the temperature later obtained in the high-pressure coils.

The experience of A. W. Peake,^a engineer in charge of gasoline production of the Mid West Oil Co., causes him to "believe that the coil area of the intercooler should be larger than the area of the aftercooler coils, as it permits condensation of more gasoline in the intercooler and reduces the chance of carrying condensate over into the high pressure cylinders, causing trouble by cutting the lubricating oil and thus wearing out the cylinders in a short time. This trouble has been experienced in quite a few plants. Increasing the intercooler area has been known to help overcome this, as also has been the placing of a steam or oil trap or some similar arrangement in the gas line, between low-pressure accumulators and high-pressure cylinder intake."

PERFECT COOLING.

Perfect cooling between low and high stage compression implies cooling the gas before it enters the high-pressure cylinder to the same temperature that it had on entering the low-stage unit. Such cooling is necessary if the two stages are to use the same amount of power in compressing equal quantities of gas an equal number of compressions. As shown in Table 3, the temperature of the gas

^a Peake, A. W., Personal communication.

at the low-stage intake is usually higher than at the high-stage intake. Such a condition would, if the number of compressions were equal in each stage, cause an unbalancing of power in the high and the low pressure cylinders. In compression plants with imperfect cooling between stages the work in the cylinders is allowed to take care of itself in such a way that the number of compressions in the two stages is not exactly equal. As stated in previous paragraphs, it would be better practice to cool the gas at the plant intake to as low a temperature as practicable in water-cooled coils and to use the same temperature between compression stages and in the high-pressure coils. In general, to keep the gas at the lowest practicable temperature at all times during treatment is the best practice. From the high-pressure coils and accumulators the gas passes to the field fuel lines or, if expanded gas is used to reduce the temperature still lower, it is led to the high-pressure coils cooled by expanded gas.

USE OF EXPANDED GAS FOR COOLING.

TABLE 4.—Data on cooling of gas by expansion at various plants.

Plant No.	Discharge temperature of gas cooled in high-pressure coils.	Method of expansion.	Expansion unit discharge.			
			First stage.		Second stage.	
			Temperature.	Pressure.	Temperature.	Pressure.
	° F.		° F.	Pounds.	° F.	Pounds.
1.		Expansion valve.....		25		
3.		Drilling engine.....		10		
4.	42	Expansion valve.....		10	(a)	(a)
6.	-6 to -17	2-stage expansion engine.....		64	-40	10
7.	20 to -7	do.....		30		
9.		do.....		30		5
10.		1-stage expansion engine.....		30		15
11.	10	do.....	-40	15		
12.		do.....		12		
14.	40	do.....	-20	25		
16.	50	do.....		14		
17.		do.....		(b)		
19.	40	do.....	32	20		
20.		do.....				
21.		do.....	32	55		
30.		Expansion valve.....				
32.	65	2-stage expansion engine.....			-30	10
33.	-30	do.....		60	-12	5
51.		Expansion valve.....				
56.		do.....				
76.	38	1-stage expansion engine.....	21			
77.	40	do.....	37	10		
78.	68	do.....	30			
79.	60	Expansion valve.....				

• Changed to expansion engine.

• Vacuum.

METHODS OF EXPANSION.

At many plants the gas after treatment in high-pressure water-cooled coils and accumulator tanks is further cooled, still at maximum pressure, in heat interchangers or double-pipe condensers by expanded gas. Two methods are used to obtain low temperatures

by the expansion of gas, (1) expanding the gas through a small opening or valve to a lower pressure, thus producing the absorption of heat, and (2) expanding the gas adiabatically in the power cylinders of a steam power unit, such as a compressor, pump, or drilling engine. Table 4 gives methods of expansion used at plants visited, also temperatures and pressures of the gas at the different stages of expansion and cooling.

EXPANSION THROUGH AN ORIFICE OR VALVE.

In the first method the high-pressure gas carrying some gasoline not removed by previous treatment is passed through either the inside or outside pipe of a double-pipe heat interchanger, and the dry gas is expanded through a small opening, such as a $\frac{1}{4}$ -inch valve, between the pipes, thus cooling the high-pressure gas and causing further condensation of gasoline. The high pressure gas and the condensate are led to an accumulator tank, where the condensate is collected and removed. From the accumulator tank the gas passes through the expansion valve of the heat exchanger and the pressure is lowered to 10 or 15 pounds, or the pressure desired or necessary to carry the gas through the field lines. The refrigerating effect obtained by this method is surprisingly small, and although many coils using this principle for cooling have been installed, few of them lower the temperature of the high-pressure gas enough to be of material benefit. In certain standard installations one coil of this type is installed with each high-pressure unit, the inside pipe of the coil having a diameter of 3 or 4 inches, and the outside 6 or 8 inches. The length is usually approximately 80 feet, and either the straight-line or return-bend type is used. After leaving this coil the gas is returned to the field for use on the lease, or sold to commercial gas companies.

COILS USED IN CONJUNCTION WITH EXPANSION ENGINES.

A study of Table 5 will give the reader an idea of the great variety of types and sizes of coils used as heat exchangers or refrigerators in conjunction with expansion engines and valve expanders. The principle is the same in all types. The cold expanded gas passes through one of the pipe members of a double pipe-interchanger while the high-pressure gas from the water-cooled coils passes through the other member. At the end of the coil in which the high-pressure gas is treated, an accumulator tank or drip collects the condensed vapor as in the water-cooled coils.

Of the plants listed in Table 5, No. 6 used the smallest size of pipe to form the double-pipe coil, 2-inch pipes inside of 3-inch.^a The cooling effect in this coil is rapid, but a small quantity of moisture in the gas will tend to freeze the coil and stop the flow, necessitating

^a The 1 $\frac{1}{2}$ -inch in 2-inch coil mentioned in Table 5 as being part of plant 11 has been abandoned in favor of a larger double coil.

continual watchfulness and more or less thawing to do. This particular coil is protected to some extent from water vapor by first passing the gas through two other double coils, 4-inch pipe in 6-inch, which are cooled by the expanded gas from the small 2-inch in 3-inch coil. In the larger coils the gas is cooled only to such a temperature that the greater part of the liquid condensed is water. The cooling area varies between 0.15 square foot and 3.70 square feet per 1,000 cubic feet of gas treated, and averages 0.563 square foot.

TABLE 5.—Data on double-pipe coils or heat exchangers used at various plants.

Plant No.	Type of coil or exchanger.	Number of coils.	Size of pipe.			Length of single coil.	Total length of coil.	Radiating area.	
			Number of pipes.	Inside diameter.	Outside diameter.			Total.	Per 1,000 cubic feet of gas treated daily.
				Inches.	Inches.	Feet.	Feet.	Sq. ft.	Sq. ft.
1	Tubular	2	50	2	36	18		940	0.940
2	Straight line	1	1	8	16	80	80	167	.372
4	do	2	1	8	12	60	120	250	.200
6	Return bend	2	1	4	6	50	200	210	.590
7	do	120	1	2	3	20	2,400	1,257	
9	Special straight line	14	5	2½	12½	100	1,400	4,575	.610
10	Straight line	5	1	8	12½	100	500	1,050	1.05
10	Special	2	1	12½	24	40	80	262	.349
11	Straight line	3	1	4	6	100	300	315	
12	Return bend	96	1	1½	2	20	1,920	630	.630
12	do	6	1	4	6	80	480	502	.714
14	do	2	1	2½	5	80	160	105	.272
14	do	2	1	4	6½	80	160	167	
16	Return bend	2	1	4	8	90	180	190	.750
17	do	2	1	8	8	80	160	130	.173
19	Horizontal tubular	4	52	1½	30	16	832	326	.181
20	Vertical tubular	2	72	2	30	12	1,728	900	.600
21	do	1	72	2	30	12	864	450	.450
30	Return bend	6	1	2	4	40	240	126	.168
32	do	20	1	2	5	40	800	630	.210
33	do	4	1	4	6	100	400	420	.233
51	do	1	2	2	10	40	40	42	.170
55	do	3	1	4	6½	80	240	189	.150
76	do	12	1	4	8	40	480	500	.250
77	do	4	1	4	8	40	160	167	.420
78	do	4	1	4	8	40	160	167	3.70
79	do	1	1	4	6	80	80	84	.330

From the 2-inch in 3-inch coil the sizes range through nearly all possible combinations up to 12½-inch inside of 24-inch. As the pipe sizes become larger, cooling is slower. The cooling effect in the last-mentioned coil is so sluggish that it is doubtful whether the coil can be considered efficient. The tubular type of heat interchangers, such as are used in plants 19, 20, and 21, either horizontal or vertical, are built in the form of a tubular boiler, the cold gas being either in the main drum shell or in the tubes. This type of interchanger has not been as satisfactory as some of the double-pipe coils.

It seems that the length of time and the necessary intimate contact between the gas and the tube surfaces is not obtained, and radiation is incomplete, the high-pressure gas being discharged at too high and the expanded gas at too low a temperature. Both the high pressure and the expanded gas is thought to follow channels through the drum

and tubes and form eddies or dead spaces, leaving some parts of the tubes and shell inactive. In this type of interchanger a series of baffles in both the tubes and shell might give the desired result, as has been done in refinery practice. This type of cooler is used at a number of plants as a water condenser and as a unit in conjunction with double-pipe coils of smaller size.

The coil used at plant 7 (see Table 5 and fig. 7) is of special straight-line construction, consisting of five $2\frac{1}{2}$ -inch pipes inside of a $12\frac{1}{2}$ -inch casing 100 feet long. This type of interchanger is not uncommon in refinery practice, being used for the interchange of heat between oil coming from and going to stills, but in the compression-plant industry its adaptation is unique. There are 14 units of this type in the entire battery. The high-pressure gas from the water-cooled coils is divided by a header and passes in parallel through the

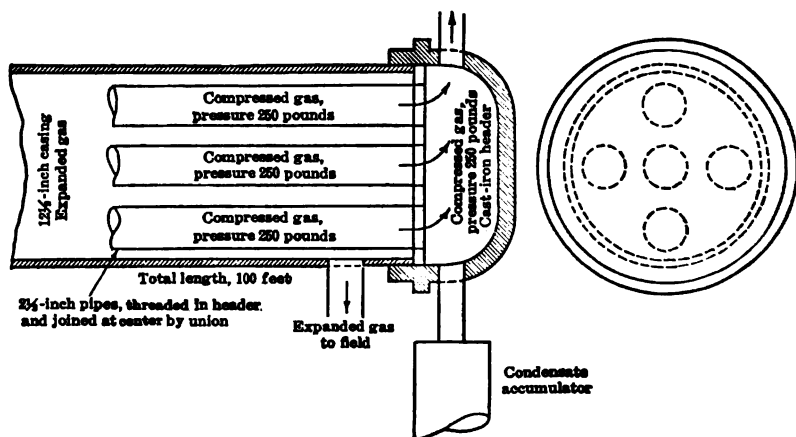


FIGURE 7.—Sections of special straight-line double coil.

$2\frac{1}{2}$ -inch pipes of four of the units, at the end of which it is collected by a header, and again divided in the same way, passing in parallel through the $2\frac{1}{2}$ -inch pipes of the remaining 10 units of the battery of 14. At the end of each unit a drip (see fig. 7) collects the condensate formed.

From the discharge header at the end of the group of 10 exchangers the gas goes directly to the expansion engine. The engine exhaust, or low-pressure cold gas, is returned to the 10 interchangers, passing through them in parallel in the $12\frac{1}{2}$ -inch shell or outside pipe counter-current to the high-pressure gas. After passing through the 10 units the gas is collected in one main or header and is again divided and passes in parallel through the outside casings of the other four units, again countercurrent to the high-pressure gas.

By using the 14 interchangers in two sets or batteries the gas is cooled in two stages, and by the counterflow system the coldest expanded gas is brought into contact with the coldest high-pressure

gas, thus making a gradual and complete interchange of heat. The expanded gas, on leaving the battery of four coils, has been raised to approximately the temperature of the high-pressure gas from the water-cooled coils, and the high-pressure gas traversing the inside pipes is brought to the lowest possible temperature by being circulated in contact with the coldest expanded gas after it has radiated a considerable portion of its heat to the partly warmed, expanded gas in the unit of four interchangers. Of the total plant production 10.4 per cent is credited to this system of cooling. These coils are not protected by a building, but are set well above the ground and housed in wooden boxes filled with sawdust, which covers the outer pipe fully 12 inches on all sides.

Plates VIII, A, and IX show views of the coils and expansion sets taken at a plant which the writer was fortunate enough to visit shortly after the installation of the expansion engine and before the cork insulation of the pipes had been completed.

INCREASE IN PRODUCTION DUE TO COOLING BY EXPANSION.

In Table 6 are recorded the percentages of total production and the gravities, in °B., of the fractions of condensate collected at the different stages in accumulator tanks in plants using expansion engines and keeping records of such data. The table shows that the percentage of condensate credited to expansion units varies between 10 and 50 per cent, except at plant 76, at which all of the condensate is collected in the accumulator tank after the cooling by expanded gas is completed. This case was cited before, and is undoubtedly due to the fact that the gas is forced to travel upward through the water-cooled coils, whereas the natural flow of any condensate formed would be downward. Apparently the condensate is absorbed by the gas and precipitated later in the double-pipe interchangers cooled by expanded gas.

TABLE 6.—Percentages and gravities of condensate collected in various accumulator tanks in plants using expansion engines.

Plant No.	Condensate produced from—					
	Low-pressure coil.		High-pressure coil.		Expansion coil.	
	Per cent.	Gravity, °B.	Per cent.	Gravity, °B.	Per cent.	Gravity, °B.
3.....	33		60	79	7	95
4.....	15		60		25	
4.....	17.6	63.8	32.7	78	a 49.7	96
6.....	24.6	60	65	65	10.4	
7.....	26	60	42	80	22	90
9.....		67		84		93
11.....	30	57	50	71	20	80
17.....					10-15	95
22.....					25	100
23.....	40	70	35	80	a 100	76-82
76.....						

a No condensate in water-cooled coils.

FIELDS IN WHICH EXPANSION UNITS ARE USED.

In the eastern fields, where much of the gas treated contains little or no fixed or true gases, being almost entirely composed of vapors of the higher hydrocarbons, the use of extremely low temperatures is of little or no benefit, and expansion engines and coils have not, in any plant known to the writer, been installed.

Mid-Continent practice, taken as a whole, does not include expansion engines as a part of the usual installation; expansion sets are, however, being adopted by some operators building new plants at the present time as a part of the original plant design. Two plants (Nos. 32 and 33) visited by the writer in Oklahoma had expansion engines and coils in service, both being in the Glenn pool. The operators of these plants stated that of the total production 10 to 25 per cent was directly due to cooling by gas expanded in engine cylinders, as shown in Table 6. The cooling area of the double-pipe heat interchangers used in the two plants averages 0.22 square foot per 1,000 cubic feet of gas treated, which is less than one-half the average area used in California practice.

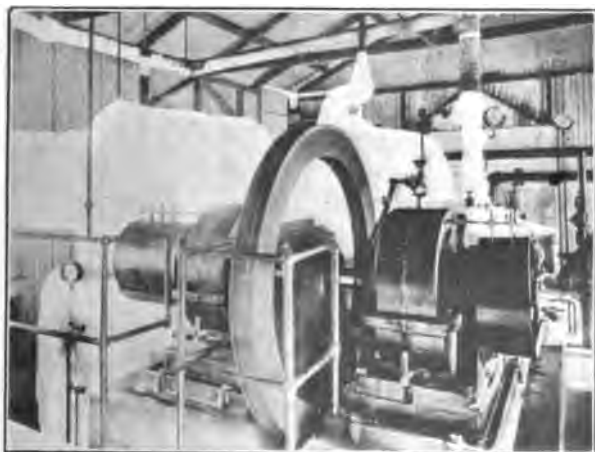
In the Caddo, Louisiana, field one company operating three compression plants uses expansion units and coils in each plant.

EXPANSION ENGINES IN CALIFORNIA FIELDS.

The widest range of development in the installation of heat interchangers and the use of expansion engines is in California practice. At plants 3, 4, 6, 7, 9, 11, and 17 in the various California fields, 7 to 50 per cent of the total production is from expansion units, as shown in Table 6. It is generally figured in these fields that 25 per cent of the production is due to the expansion treatment. A plant in the Fullerton field treating 2,500,000 cubic feet of gas daily produced 3,000 to 3,200 gallons of condensate with a gravity of 72° B. before the expansion unit was put into operation. The expansion unit increased the daily production to between 4,000 and 4,300 gallons of condensate with a gravity of 80° to 84° B., or about 25 per cent of the total yield. The radiating areas in the heat exchangers used in California plants (see Table 6, plants 1 to 21) vary from 0.20 to 1.05 square feet per 1,000 cubic feet of gas treated daily. A radiating area large enough to warm the cold or expanded gas to the temperature, as nearly as practicable, of the high-pressure gas from the water-cooled coils is all that is necessary, because any further increase in cooling surface, the two gases being brought to approximately the same temperature, has no effect. The proper area for each 1,000 cubic feet of gas to be treated daily is a factor that can be obtained only by experiment at each plant, unless the designer has had experience in the particular field and with the particular gas

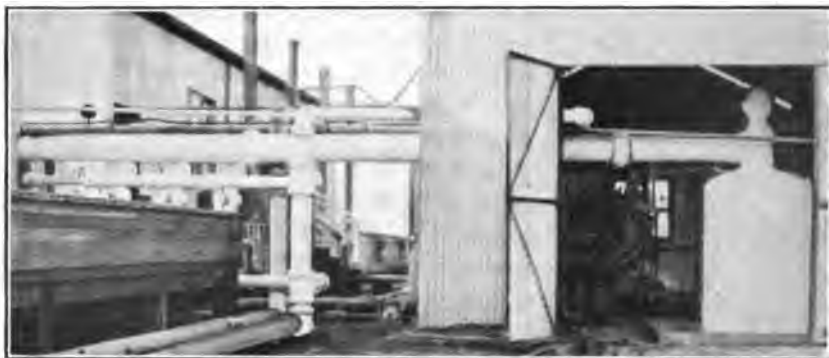


A. END OF HORIZONTAL TUBULAR COOLER, TYPE USED IN PLANTS 19, 20, AND 21.

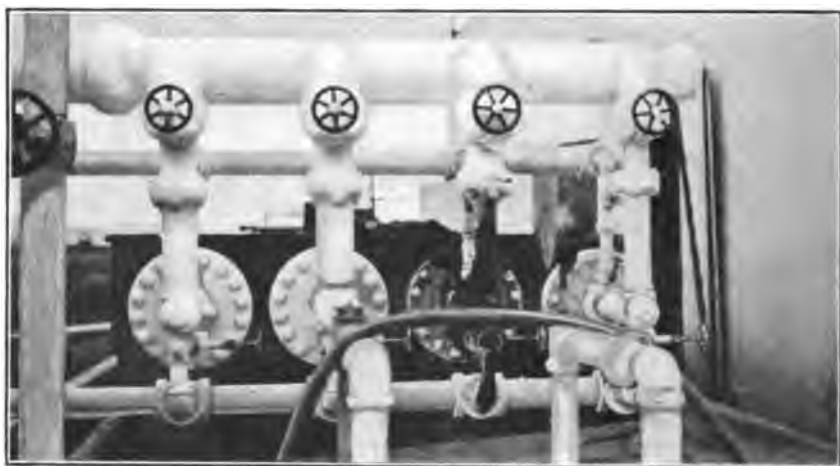


B. TWO-STAGE EXPANSION ENGINE IN COMPRESSION PLANT.

Note ice on accumulator tank and piping.



A. EXPANSION-EXHAUST ACCUMULATOR TANK AND PIPING. TANKS SHOW 2-INCH COATING OF FROST. TEMPERATURE OF GAS WAS BELOW 0° F.



B. END OF DOUBLE-PIPE COILS AND INTAKE MANIFOLD FOR EXPANDED GAS.



C. HEADER FOR EXPANDED GAS. CORK INSULATED INTAKE, PIPES EXPOSED TO AIR.

to be treated, or has data on the area used and the efficiency obtained at another plant already at work in the same field and treating the same gas.

TEMPERATURES OBTAINED FROM GAS EXPANSION.

The temperatures produced by heat exchange on high-pressure gas vary in practice from -17° to $+65^{\circ}$ F., or between 75° below and 20° F. below atmospheric temperature, as shown in Table 4 (p. 41). The best temperature to use in a given plant depends, as does the ultimate high pressure used, on the characteristics of the gas and the product desired, also on the efficiency of extraction and temperature previously obtained in the water-cooled coils. As the condensation of vapors depends on both temperature and pressure, in accordance with the physical laws of gases, at a given pressure there is some critical temperature below which it is useless to cool the gas, as Burrell^a has demonstrated in the laboratory method of determining the quantity of condensable vapors in any given gas. The method of laboratory test uses only atmospheric (14.4 pounds at an elevation of 600 feet) pressure, and the extremely low temperature of 115° C. below zero, which is equal to 175° F. below zero.

At this temperature all of the propane and the butane are liquefied, which in compression practice is neither practicable nor desirable, because these two hydrocarbons are so volatile at atmospheric temperatures and pressures as cause them to weather out of plant products to a large extent, if not entirely. As a portion of each condensable hydrocarbon is precipitated and taken out of the gas, the pressure necessary to condense the remaining portion at a constant temperature rises, in accordance with the law of partial pressures. Or, with the pressure remaining constant, the temperature must be reduced to precipitate the remaining portion of that particular vapor fraction. Conversely, if at a given pressure any condensable constituent would be entirely precipitated when the gas reached the critical temperature of that fraction, the composition of the gas would be simplified and the precipitation of the other condensable fractions more complete at the same pressure. This has been demonstrated in practice in a plant using a maximum pressure of 250 pounds, and a temperature as low as 10° F. below zero at the discharge of the high-pressure coil, which was cooled by expanded gas. (See Pl. IX, A.) The condensate collected in the tank at the end of the coil contained more than 1 per cent of water; a small percentage of water was also found with the lightest condensate precipitated at the exhaust of the second stage of the expansion engine. (See Pl. VIII, B.) If water vapor is retained through all of the steps of compression and

^a Burrell, G. A., and Jones, G. W., *Methods of testing natural gas for gasoline content*: Tech. Paper, 87, Bureau of Mines, 1916, p. 26.

cooling, as demonstrated in the plant just cited, it is probable that portions of all the hydrocarbon fractions are also. The points to be determined in any plant are the quantity of such vapors being lost and the temperature necessary to condense them as well as the value of the product and the cost of an installation to obtain the required results.

The value of the condensate obtained will depend on the quantity that can be marketed, and if excessive pressures are used, the product may be so light and volatile as to be of little value. A proper relation between pressure and temperature will in all instances yield the maximum marketable condensate from any given gas, and this relation can be found only by trials and tests made at each plant. In the plant cited above, the product obtained in the accumulator at the expansion engine exhaust had a gravity of 105° B., and probably consisted principally of butane, with small proportions of the other higher hydrocarbons and of water. The gas in this accumulator had a temperature of 40° F. below zero and a pressure of 10 pounds, as the result of the two stages of expansion. It was found that in the expansion cylinder of the engine, the gas had reached a temperature lower than 100° F. below zero before being exhausted. The quick rise in the temperature of the gas between the cylinder and the tank, which was connected to the exhaust by a short insulated pipe, is probably due to two factors, radiation from the cylinder walls and the latent heat of vapors given out on condensation.

In a plant in Oklahoma, in experimenting with expansion units, it was found that the desired low temperatures could not be obtained. So little vapor was precipitated in the coils ahead of the expansion unit that in expanding the gas the latent heat given up by the condensing vapors immediately reheated the expanding gas to 20° F. The remedy for a condition of this kind would necessarily have to be found either in the pressure or water cooling to which the gas was being subjected.

TEMPERATURE AND PRESSURE CHANGES IN A COMPRESSION PLANT.

Figure 8 shows diagrammatically the changes in both pressure and temperature to which gas is subjected in average compression-plant practice. The vertical coordinates show temperatures in degrees Fahrenheit for the solid line or temperature curve, and pressures in pounds per square inch for the dotted or pressure curve. The horizontal coordinates represent the different stages of treatment at which the changes of pressure and temperature occur.

FLOW SHEET OF A COMPRESSION PLANT.

Figure 9 shows the gas flow diagram of a 2-stage compression plant using single-stage expansion, connected with two sets of expanded-gas cooled coils in series. The gas intakes and discharges of the com-

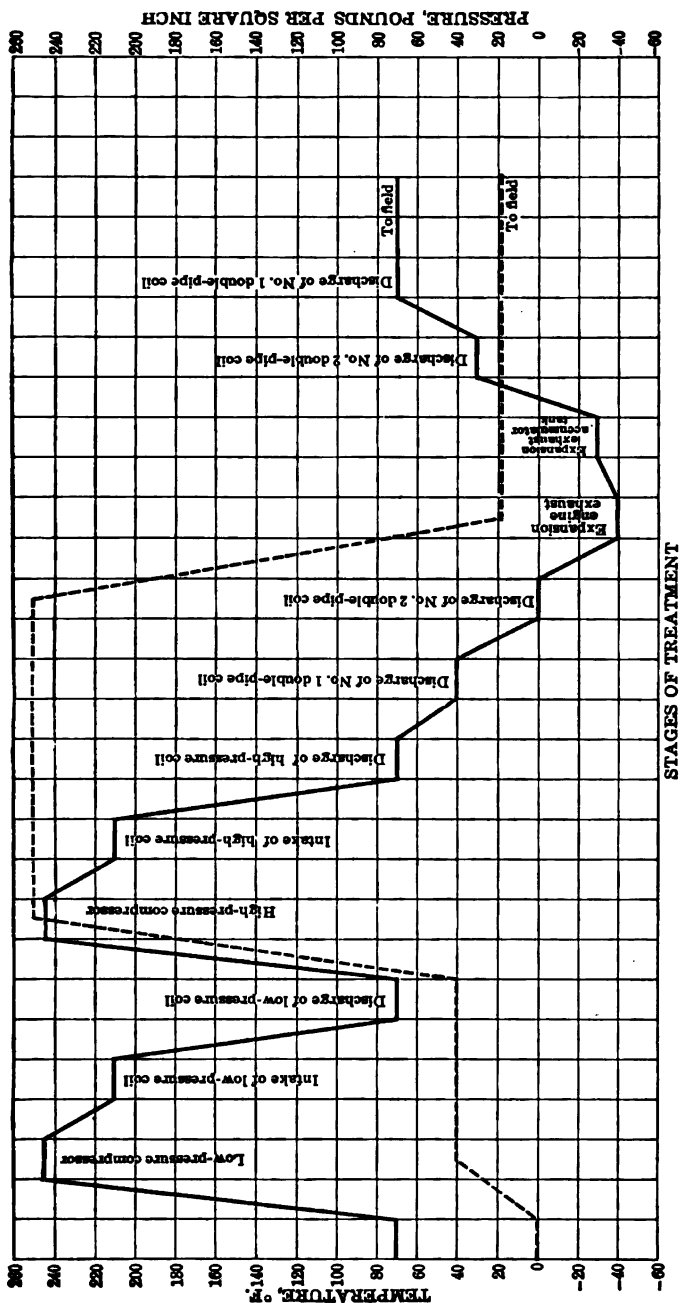


FIGURE 8.—Diagram showing the changes in temperature and pressure of gas in compression-plant practice. Dotted line is pressure curve, solid line is temperature curve.

pressors and the water-cooled coils are shown connected in manifold, this system of connections being the most flexible. With valves properly placed, any unit, part of a unit, or coil may be cut out for

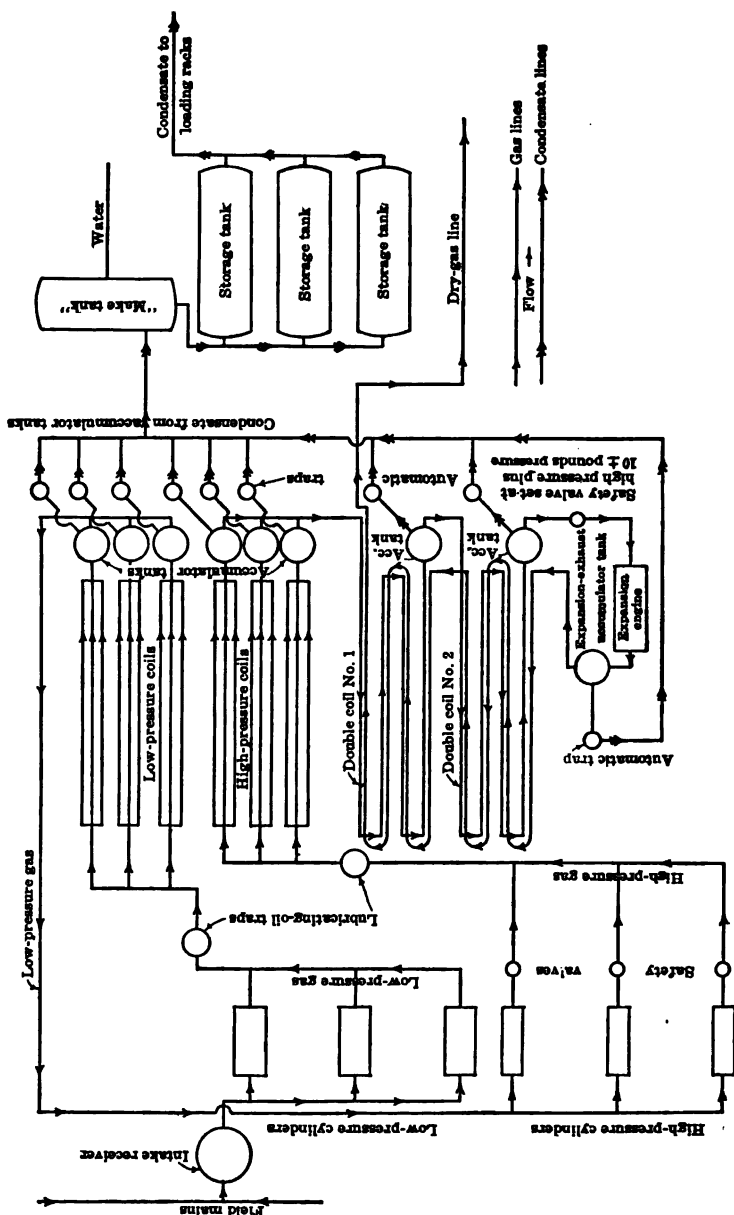
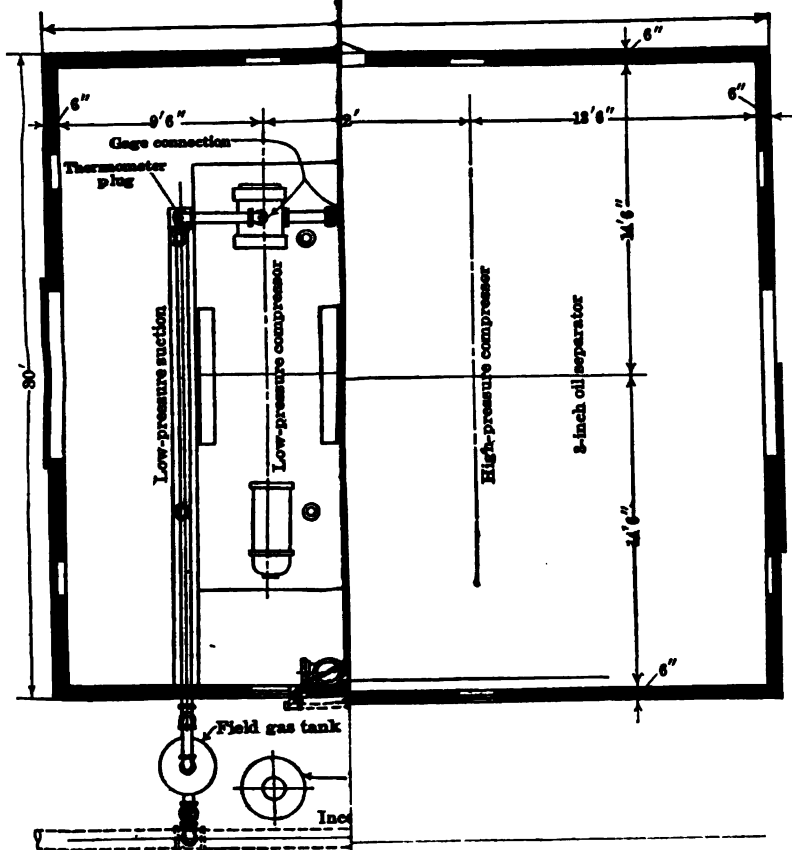
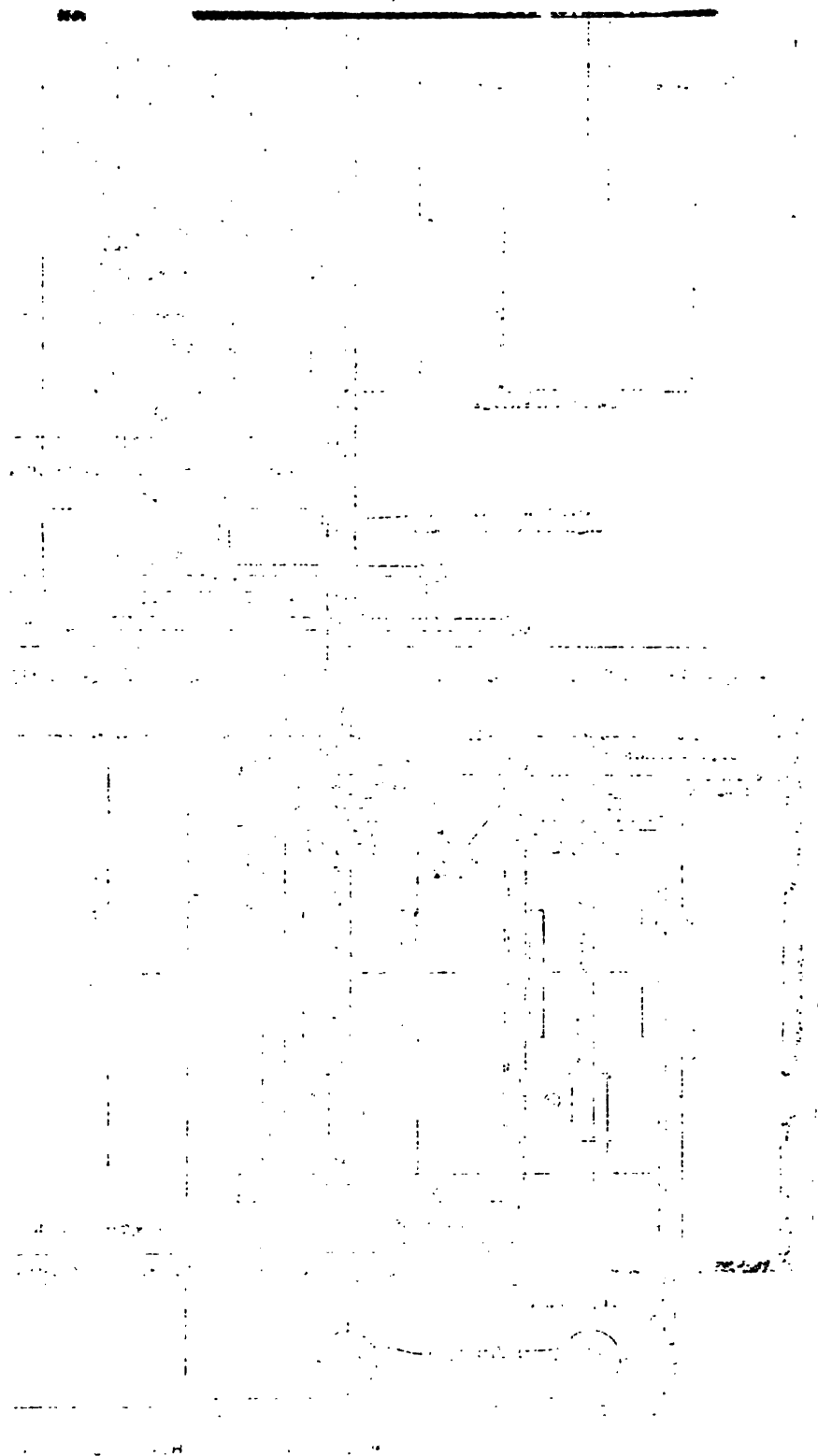
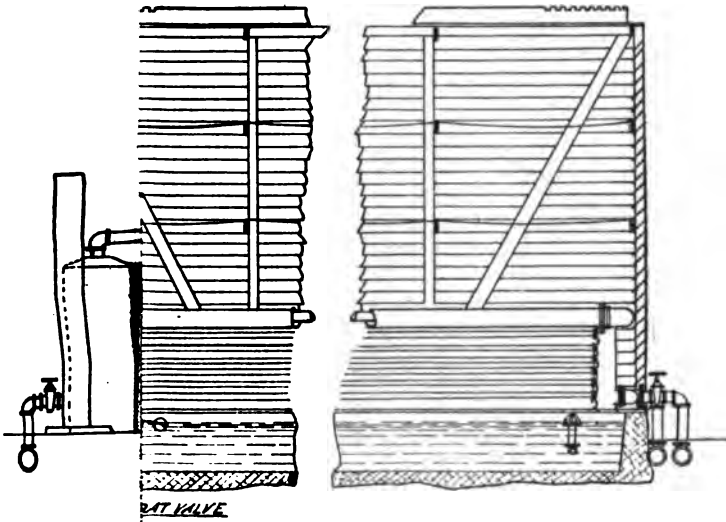


FIGURE 9.—Flow sheet of compression plant using 2-stage compression and single-stage expansion.

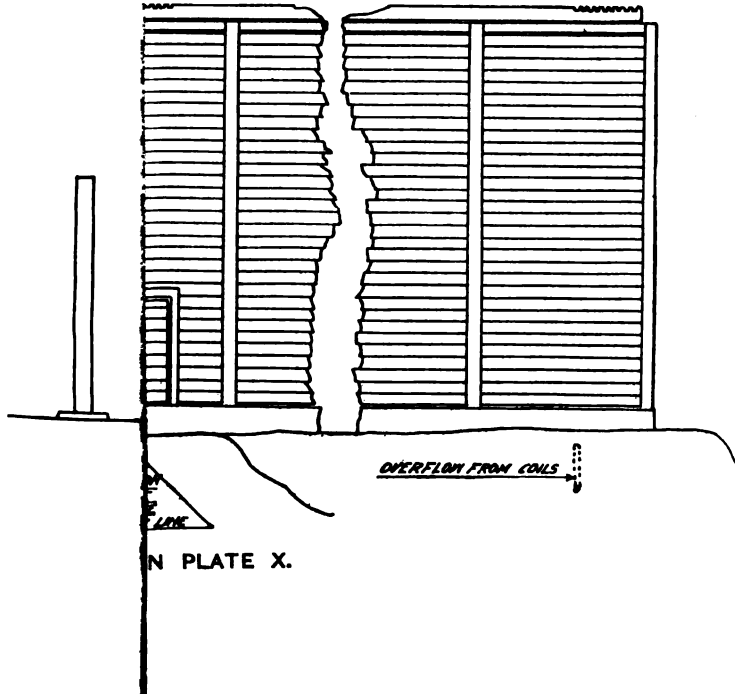
inspection or repairs without shutting down any other part or stopping plant operations, the work of the inactive unit being taken up by a small overload on the other units.







COOLING TOWER



A single-stage expansion engine is shown connected with two double-pipe coils in series; if two stages of expansion were being used each of the double-pipe coils would use the gas expanded in one stage and be so connected that the gas from coil No. 1, using first-stage expanded gas, would return to the second-stage expansion cylinder, from which it would be returned to coil No. 2 to further cool the high-pressure gas.

Plates X and XI show plans and elevations of a typical direct-connected compressor plant. The low-pressure and the high-pressure cylinders are operated independently by separate gas engines.

USE OF POWER DEVELOPED BY GAS EXPANSION.

At all plants where expansion units are used, the development of power by the expansion of the gas has been a secondary consideration. In a number of plants power is developed only to give resistance to the expanding gas, as in a plant using a pump under a back pressure of 150 pounds, and in another plant compressing air to 40-pound pressure, only to release it to the atmosphere through a small valve set to hold the required back pressure on the natural gas.

In plant 17 the compressor end of the expansion unit holds a vacuum on the double-pipe coil and on the exhaust of the power cylinder, and delivers the gas at a pressure sufficient to return it to the lease. (See Pl. XII, A.) This use of the power developed is not uncommon, although usually the compressor suction is above atmospheric pressure. At plant 76 the power developed from expanding the gas in two single-stage cylinders working in duplex is used to compress air in two stages to a pressure of 85 pounds, to be used in air lifts in pumping oil wells. Other plants use the power generated by single-stage expansion units, as in the tandem compressor, or by two-stage expansion units, as in the cross-compound machines, to drive one of the two-stage compressor units connected in parallel with the other compression cylinders, both at the intake and the discharge. A compressor used in this way may, as at plants 6 and 14, take gas from and deliver it to the manifolds used for the intake and discharge of the other compression units.

EFFICIENCY OF EXPANSION UNITS.

The amount of power developed in an expansion engine as compared with the amount used in compression is very low, probably not more than 5 or 10 per cent in average plants. This condition is to be expected because of the energy loss through dissipation of heat, the consumption of power in operating the piston and valve in the expansion cylinder, and the reduction in pressure through losses of gas and vapor by condensation and leakage during transmission through pipes and cooling systems.

An exceptional instance of power developed in an expansion unit was noted at a California plant. Tests showed that between 25 and 35 per cent of the power used to compress the gas was developed by the expansion engine, which used in its power cylinders all of the gas compressed in two stages of expansion, with heating between the two expansion cylinders. The plant compressed 1,900 cubic feet of gas per minute, of which 690 cubic feet was compressed in the compressor end of the expansion engine.

The gas entering the high-pressure cylinder of the expansion engine at a pressure of 250 pounds had a temperature of 5° F. below zero; the discharge pressure was 50 pounds. The gas was then passed through double-pipe coils and heated to 50° F. before it entered the low-pressure power cylinder. From this cylinder it was exhausted at a pressure of 10 pounds and a temperature of 40° F. below zero. The gas from the second expansion exhaust was used for cooling the high-pressure gas, and was then discharged from the plant at 57° F. A measurement of the temperature in the low-pressure expansion cylinder indicated a temperature of 140° F. below zero.

If a more efficient return of power were made an object, more power could be developed by heating the high-pressure gas, after it leaves the double-pipe interchanger, in an interchanger with the hot gas from the high and the low pressure compression cylinders. If desirable the gas could be further heated in a double-pipe interchanger by the exhaust from the power cylinder of the gas engine, or in a tubular interchanger such as is used for preheating boiler water with exhaust gases.

Heating the compressed gas before it enters the valve chest of the expansion cylinder would also tend to reduce freezing at that point, thus reducing the power used in moving the valve mechanism.

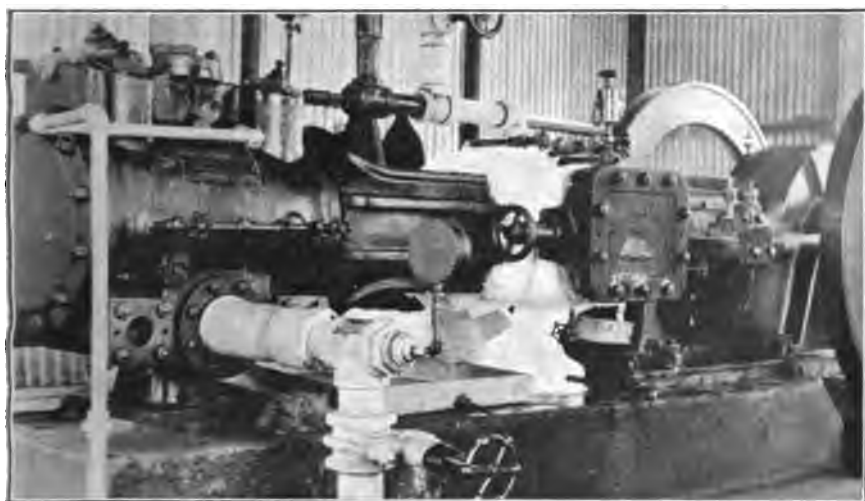
To obtain as low temperatures of the expanded gas, with preheating as without preheating, more power would have to be developed and used by the expansion unit. This could be accomplished by added expansion units, by increasing the load, or by running the compressor faster at the same pressure, thus increasing the total volume of gas compressed in a given unit of time.

EXPANSION FOR POWER ALONE.

In expanding compressed gas for power purposes only, and not for refrigeration, as is contemplated in an eastern plant, preheating the gases by either the hot compressed gas or the engine exhaust would be necessary in order to make the installation a commercial success. Using a relatively small quantity of gas at a low temperature (60° F.) would hardly pay in power delivered, as compared with a gas engine using gas worth 15 cents or less per 1,000 cubic feet.



A. LOADING RACK FOR TANK CARS AT BLENDING STATION.



B. SINGLE-STAGE COMPRESSOR USED AS EXPANSION ENGINE PUMPING GAS TO FIELD AT 60 POUNDS PRESSURE.



C. COMPRESSION PLANT, SHOWING HORIZONTAL TYPE OF "MAKE" TANKS IN MIDDLE FOREGROUND, AND OF ACCUMULATOR TANKS, AT RIGHT.

USES MADE OF TREATED GAS.

With the passage of the gas through the coils of the expansion unit, the treatment for recovery of gasoline is finished.

Gas used for power to run the compression plant, either in gas engines or under boilers, is universally taken from the treated gas, the rest being returned to the lease or pumped into the lines of a commercial gas company. The lease or contract usually stipulates that gas used for power in the compression plant is to be taken from the treated gas at no cost to the lessee.

GAS USED PER HORSEPOWER IN COMPRESSION UNITS.

The quantity of gas necessary to operate a gas engine will vary from 9 to 18 cubic feet per horsepower-hour, depending on the size, type, and age of the engine and the care given it. Boilers in a steam plant require 40 to 120 feet per horsepower hour. A plant having six 80-horsepower, direct-connected, horizontal, 2-cycle gas engines used between 12 and 14 feet. The engines had been in service two years and had been well cared for. The large 450 to 1,100 horsepower, horizontal, direct-connected, tandem, double-acting, 4-cycle units will use as little as 9 feet and the 150 to 200 horsepower, vertical, 4-cylinder, 4-cycle, belted units will use 10 to 14 feet per horsepower hour.

Oliphant^a gives the following table showing the average amount of natural gas required to operate gas engines or to supply a steam-engine plant using natural gas as fuel under the boilers in cubic feet per indicated horsepower hour:

Cubic feet of natural gas required per horsepower-hour to drive a gas engine or steam plant.

	Cu. ft.
Large gas engine, highest type.....	9
Ordinary gas engine.....	13
Triple-expansion condensing steam engine.....	16
Double-expansion condensing steam engine.....	20
Single-cylinder steam engine with cut-off.....	40
Ordinary high-pressure steam engine without cut-off.....	80
Ordinary oil-well pumping steam engine.....	130

From 10 to 12 cubic feet of air is necessary for the complete combustion of 1 cubic foot of natural gas.

PERCENTAGE OF GAS RETURNED AFTER CONDENSATE AND GAS USED FOR POWER IS DEDUCTED.

The reduction in the volume of the gas from treatment by the compression process is unaccountably large. As has been stated, in treating gas from old wells that have been gas-pumped for years and are held at high vacuums, practically all of the gas disappears in one stage or another of the treatment, often not enough being left to run the engines.

^a Oliphant, F. H., Catalogue of metric metal works, 1914, p. 42.

Gas produced under natural pressure in the newer fields contained a much higher percentage of fixed gases, hence the quantity of residual gas, after condensate and gas used for power have been deducted, reaches 70, or, in some plants, it is claimed, 80 per cent of the total wet gas entering the plant. Exact figures on the quantities of residual gas are seldom kept in compression plants, it being a matter of little importance in most fields. It appears, however, from the figures that the writer was able to obtain that in plants treating gas yielding $1\frac{1}{2}$ to 2 gallons of condensate per 1,000 cubic feet the net amount of gas left after treatment and deducting gas used as fuel, would average approximately 66 per cent of the total amount entering the plant. As the gas being treated increased in gasoline content the quantity of residual gas became less and less until it was not sufficient to furnish power for compression.

GAS USED FOR POWER AND TO FORM CONDENSATE.

Between 10 and 15 per cent of the total quantity of gas treated is required for power purposes, depending on the type and the efficiency of the engines driving the compressors. Burrell^a states that it takes, on an average, 35 cubic feet of vapor to produce 1 gallon of condensate, or 3.5 per cent for each gallon produced from 1,000 cubic feet of gas, or if 3 gallons of unweathered product is made it will account for 10.5 per cent of the total gas entering the plant. The condensation of water vapor during the treatment will also account for a certain percentage of the total volume. The gas and vapor unaccounted for in the ways noted must be lost by leakage in pipes and machines during the various stages of plant operation, and by weathering of light fractions.

FOREIGN CONSTITUENTS IN NATURAL GAS.

AIR.

Besides water vapor, a number of gases not of the hydrocarbon groups are often found in natural gas. Air, if present, may be a constituent of gas from wells under high vacuums, but is usually due to inward leakage, either in the well casing or in lines transmitting the gas to the plant. At a plant visited in the Shallow pool of Oklahoma, the writer was informed that the gas being treated contained 30 per cent air. The area from which the gas was being drawn covered 12 square miles, making the use of long gathering lines and many vacuum pumps necessary, which probably accounts for the extremely high air content.

^a Burrell, G. A., Seibert, F. M., and Oberfell, G. G., The condensation of gasoline from natural gas: Bull. 88, Bureau of Mines, 1915, p. 60.

WATER VAPOR.

Air always carries more or less water vapor with it, and this fact may account for part of the water precipitated with condensate. As most oils carry some water with them from the oil-bearing formations, it is natural to believe that some water vapor from this source would be carried into the gas, especially when the wells are being gas-pumped and low pressures maintained. The temperature of the oil and water under ground would also tend to allow water vapor to form and be held in the gas. In one instance, the oil coming from a certain well also producing gas had a temperature of 150° F. in the flow lines.

A California plant treating 2,500,000 cubic feet of gas daily produced water with the condensate at various points as follows:

Gallons of water drained from various points at California plant in 24 hours.

	Gallons.
From low-pressure accumulator.....	200
From high-pressure accumulator.....	50
From final double-pipe coil.....	25
From storage tank.....	65
Total.....	340

This quantity of water was equal to between 5 and 6 per cent of the condensate produced.

CARBON DIOXIDE.

The proportion of carbon dioxide in natural gas used in compression plants varies widely. As much as 30 per cent has been found in both the Mid-Continent and California fields. So large a proportion is unusual, but percentages up to 10 are not uncommon in California fields, and in some districts in Oklahoma.

Nitrogen is found in the natural gas in some districts, as noted by Burrell,^a but was not found in gas fields visited and sampled by the writer, except as introduced into the gas with air.

SULPHUR COMPOUNDS.

Hydrogen sulphide or other gaseous sulphur compounds, usually called "sulphur gas," is found in many of the fields producing casing-head gas. However, only the gas from small areas of these fields contains sulphur in such quantities as to be a decidedly detrimental factor in treatment of gas for its gasoline content.

Plants in the southern Illinois field are troubled by sulphur compounds more generally than those in any other district as a whole. A plant in the Santa Maria and one in the Salt Lake field in California report sulphur trouble, but such contamination is local and is not characteristic of those fields as a whole.

^a Burrell, G. A., Seibert, F. M., and Oberfell, G. G., The condensation of gasoline from natural gas; Bull. 88, Bureau of Mines, 1916, pp. 21-22.

GENERAL EFFECTS ON COMPRESSION TREATMENT.

The gases named, with the exception of hydrogen sulphide, are inert chemically through all the stages of the compression process. Physically they affect plant practice in two ways. They cut down the volume of productive gas treated or absorb power for which no return is possible; they complicate the problem of partial pressures by requiring higher pressures for the gas as a whole in order to bring any one of the condensable fractions to its critical pressure, thus again necessitating more power.

EFFECTS OF SULPHUR AND METHODS OF REMOVAL.

In the Lawrenceville district of southern Illinois the proportion of sulphur in the natural gas is so large as to be a decided detriment to treatment by compression. Beside destroying the pipes in cooling coils, so much of the hydrogen sulphide is dissolved in the condensate, either as a gas or as a liquid, that resort is had to steam distilling in order to free the condensate from this objectionable content. The odor of even small proportions is noticeable, making the product unsalable, and such gasoline, if used in motors, will attack the pistons and cylinders, causing pitting and roughness. By use of the steam stills in that district between one-third and one-half of the total plant product is lost as noncondensable vapor passing through the cooling coils after the stills.

The two California plants, mentioned previously, report no trouble with sulphur in the condensate, but do have trouble from the sulphur gas attacking and eating out cooling coils. At the plant in the Salt Lake field 2-inch steel pipes will often be eaten through, particularly at a low point in the coil, in 3 or 4 months. The plant in the Santa Maria field found that 2-inch steel pipe, costing 12 cents per foot (May, 1916), in the cooling towers lasted 6 months, and that wrought-iron pipes, costing 19 cents per foot at that time, lasted 13 months or more.

No compressor or engine trouble traceable to hydrogen sulphide gas was reported at any of the above plants, possibly because of the film of lubricating oil constantly protecting the pistons and cylinders.

The elimination of sulphur gas has long been a part of the treatment for purifying manufactured or artificial gas. The artificial gas made from coal or oil is passed through a scrubber containing iron oxide (common hematite iron ore). A chemical reaction takes place, the sulphur uniting with the iron to form iron sulphide, which is a solid at normal temperatures, thus removing the sulphur from the gas. When the iron becomes slow in its action, or so largely converted to the sulphide as to be inefficient in removing the sulphur, it is discharged and a fresh charge placed in the scrubber. The scrubbers are generally operated in pairs to allow one to be cut out

during periods of cleaning and charging. The discharged iron ore is thrown out on the ground, where it is oxidized by the action of the sun and the atmosphere, and again used as a fresh charge for the scrubbers.

The sulphur compounds originating in gas act chemically as an acid in much the same way as the acid fumes that are carried in still vapors in oil refining.

An eastern refinery which compresses the uncondensed still vapors and gases to further remove condensable fractions uses a series of water and caustic washes in scrubbing tanks to remove acid impurities, as described on page 68.

To overcome the action of sulphur gas which was eating out the high-pressure coils of a compression plant, Mr. D. L. Newton, general superintendent of the Hurley Smith Gasoline Co., of Los Angeles, Cal., designed and successfully operated a scrubber and cooler combined which removed the objectionable gases and also took the place of the high-pressure water-cooled coils.

Figure 10 shows the general design and method of operation of the scrubber. After this treatment the gas was refrigerated in double-pipe coils of usual construction without causing their destruction or forming incrustations of sulphur compounds.

The following data, regarding the operation of the scrubber, was also kindly furnished by Mr. Newton:

Data on scrubber for removing sulphur.

Capacity, 300,000 cubic feet per day at a pressure of 250 pounds.

Temperature of gas entering scrubber, 190° F.

Temperature of gas leaving scrubber, 78° F.

Temperature of water entering scrubber, 72° F.

Temperature of water leaving scrubber, 78° F.

Volume of water used per 24 hours, 15,000 gallons.

The water used in the scrubber, after being automatically trapped from the separator, was returned to the top of the cooling tower for cooling in the usual way. Owing to aeration and the lowered pressure, the greater part of the sulphur gas passed off into the air, the cooled water being returned to the scrubber by a pump at a pressure slightly higher than that at which the gas entered the scrubber.

CONDENSATE.

LINE DRIP.

The first condensate produced in treating gas by compression is the small quantity of rather heavy and often discolored naphtha accumulating in the pipe-line drips. After this condensate has been collected and cleaned by filtering or distilling it is mixed with the balance of the plant product, giving the mixture a lower gravity and vapor tension and helping to stabilize the "wild" condensate from other parts of the plant.

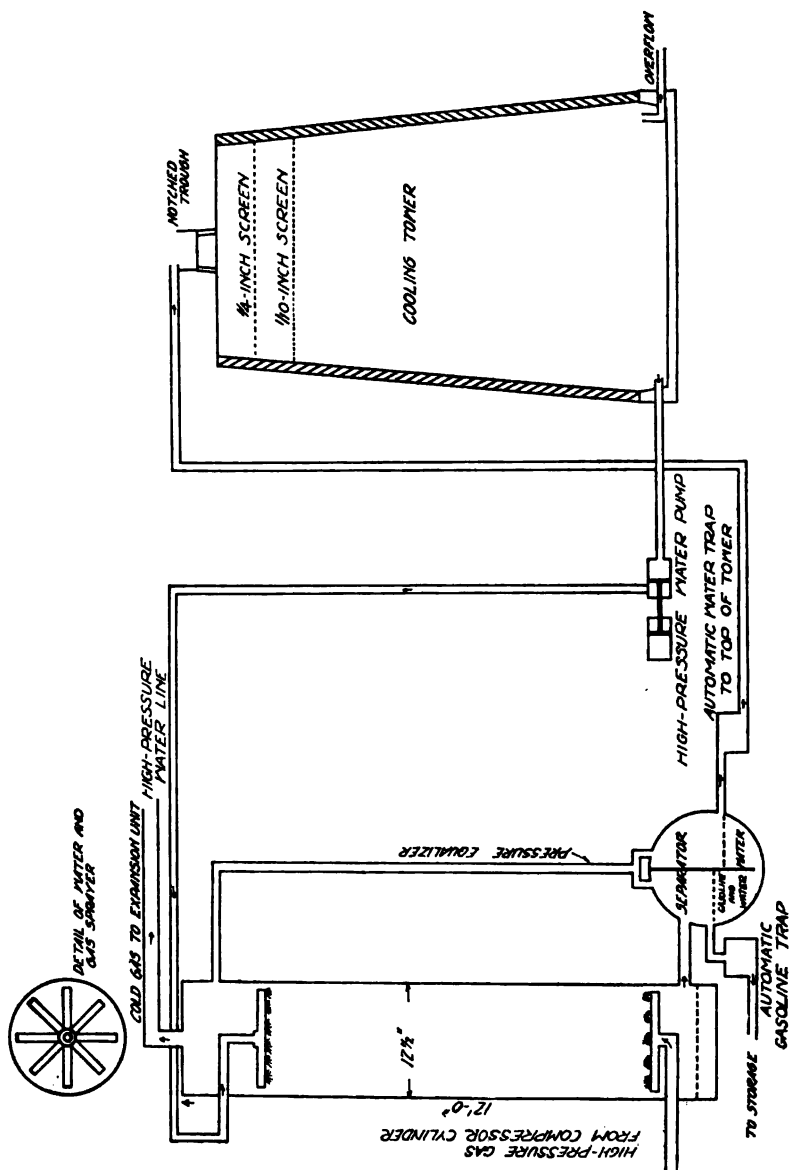


FIGURE 10.—Scrubber used to cool gas and remove sulphur compounds.

CONDENSATE FROM LOW-PRESSURE COILS.

The second condensate is that collected in the low-pressure accumulator tanks; its proportion to the whole product varies between nothing and 50 per cent, and in average plant practice between 10 and 30 per cent. The gravity varies around 60° B. and the vapor tension between 1 and 3 pounds. Such condensate makes an ideal motor fuel just as it comes from the coils, but is usually mixed before leaving the plant with the other products either in "make tanks" or in storage tanks, having the same tendency as the line drip to improve the product as a whole.

The following distillation represents approximately an average first-stage condensate as produced in California practice:

Results of fractional distillation of first-compression naphtha (condensate).

[Analyst, Paul W. Prutzman.]

Still charged with 500 c. c., at 62° B.; started over at 106° F.

Cut No.	Amount of cut.	Final temper- ature.	Gravity of cut.	Per cent.
	c. c.	°F.	°B.	
1.....	50	174	76.8	10
2.....	50	192	71.8	10
3.....	50	206	66.5	10
4.....	50	218	63.1	10
5.....	50	228	60.8	10
6.....	50	241	58.4	10
7.....	50	254	56.5	10
8.....	50	272	54.6	10
9.....	50	312	51.4	10
.....	16	352		

Total amount distilled off and collected, 466 c. c.; final temperature, 352° F. Loss, 8 per cent.

Eastern or Mid-Continent products with these end points would have a specific gravity between 5° and 9° B. higher, but the sample distilled is fairly representative of the first-stage product of two-stage plants.

CONDENSATE FROM HIGH-PRESSURE COILS.

Next in the series of condensates collected is that obtained from the gas under the maximum pressure used in any given plant and at temperatures developed by water cooling.

In average plant practice the condensate precipitated and collected at this point represents the principal bulk of the total recovery, seldom being less than 30 per cent of the total product even in plants using expansion units, and at times reaching 100 per cent, as in all single-stage practice and in some two-stage installations. At plant 22, which compressed the gas to 300 pounds in two stages and cooled it to 60° F. in the high-pressure coils, all the condensate was produced at this point, the product having a gravity of 96° B.

The specific gravity of the high-stage condensate is between 65° B. and 100° B., averaging in eastern fields approximately 85° B., in Oklahoma 78° B., and in California 72° B.

As formed in the accumulator tank this condensate is "wild," owing to the absence of low-gravity fractions, to dissolved gas, and to hydrocarbons that can be held as liquids only under high pressure at the temperatures attained in the water-cooled coils. As the pressure is reduced by the automatic traps or in the transfer from accumulators to the "make" or storage tanks, the lighter fractions and dissolved gases immediately start coming off and build up pressure in the tank containing them or escape to the atmosphere. For these reasons the condensate from high-pressure accumulators is usually discharged to tanks containing the heavier fractions precipitated in other coils and is often blended in there, or before it reaches the storage tanks.

CONDENSATE FROM EXPANSION COIL.

Table 6 (p. 45) gives the gravity and percentage of this condensate as obtained in plants using expansion units. The condensate has much the same physical characteristics as the condensate from high-pressure water-cooled coils and is handled and treated in the same way.

CONDENSATE FROM EXPANSION EXHAUST.

As stated and discussed under expansion units, plant 6 collects a high-gravity (105° B.) condensate in an accumulator close-connected to the exhaust of the second-stage expansion cylinder. This condensate is stored separately and held under pressure until blended with large quantities of 48° B. naphtha.

At the plant shown in Plates VIII, A (p. 46), and IX, A (p. 47), the condensate collected in the expansion-exhaust accumulator tank is mixed with the balance of the plant product and the mixture is shipped, without blending, by auto trucks.

CONDENSABLE HYDROCARBON FRACTIONS IN NATURAL GAS.

From these data and from points previously brought out, it appears that different natural gas from different fields containing the same quantity of condensable vapors seldom contains the same percentages of the various hydrocarbon fractions entering into the composition of gasoline.

This, in part at least, explains the wide variation in the gravity and amount of product obtained under similar conditions of temperature and pressure in different plants treating gas from different parts of the same field or from different fields.

From the study made by the writer of the conditions as found in fields throughout the United States, the explanation seems to lie in

any one, or a combination, of three conditions—the fractional composition of the gasoline content of the oil from which the gas comes, the temperature of the oil, and the pressure on the oil at the time of releasing the vapors to the gas.

Under the first condition, as demonstrated by refinery results and fractionations, it has been shown that distillates of the same gravity and the same or different end points vary widely in the content of the different fractions obtained between the same temperature limits. The oil containing only certain fractions of the light hydrocarbons can give up only these fractions to the gas, regardless of the temperature or pressure.

As the temperature of the oil varies in different fields and at different localities and depths in the same field, the factor of temperature must bear directly on the fractions of distillates contained in the gas. At constant pressures and temperatures only certain of the fractions will vaporize and be carried into the gas, leaving other fractions as liquids with the oil.

In any oil field, rock pressures decline as the supply of gas and oil is reduced. As the boiling or vaporizing temperatures of liquids are lowered by reduced pressures, this condition of lower pressures allows the less volatile, heavier fractions to vaporize if the temperature remains constant, thus adding to the gasoline content of the gas. As the rock pressure becomes low and the rate of decline so slow as to be practically stationary, and vapors no longer distill naturally from the oil left in the ground, resort is had to vacuum pumps to increase the flow of oil and gas, thus permitting the vapors to distill from the oil in the sands. Under this method is produced the gas of which the greatest proportion is condensable, as in eastern compression practice.

The widely varying conditions under which casing-head gas obtains its charge of condensable vapors, the variable content of lighter hydrocarbons in the oil in the ground, and the direct effects of the law of partial pressures on the products precipitated at the different stages in plant practice will to no small extent account for the great differences in plant practice as to pressures and temperatures used, and also for the variations in the percentage and gravity of the condensates of successive plant stages and the gravity and vapor tensions of the product of plants in different fields.

VARIATIONS IN PLANT PRODUCT.

The quantity and the gravity of condensates from different plants is shown in Table 2 (p. 29). These figures represent the average quantity and the average gravity, but both vary considerably from day to day and month to month. Many theories have been advanced to account for these variations, but none of them is satisfactory in analyzing the variations as they actually occur in plant production.

From the known effects of temperature on gas containing condensable vapors, it is reasonable to believe that atmospheric conditions, changing as they do with the seasons, explain in part the differences noticed from time to time in the yields from the different stages and in the total plant output. Many operators state that a larger proportion of condensate of higher gravity is collected in the low-stage accumulator tanks in cold weather. This same result is noticed also in the quantity and gravity of "line distillate" precipitated in gathering systems and collected in the line drips. In many plants the total production increases in winter, but this is not always true, as in plants using low temperatures, produced by expansion engines. It is often found that cold weather condenses some vapor in the gathering lines, and unless this is collected and added to the total an actual decrease in plant production results. It has been shown by several records of plant production that the yield decreases on a cold day and increases to a point well above the average on the first warm day after a period of cold weather. A plant in a hot climate on keeping records of the variations in production and attempting to determine the factors concerned, found that in general the production usually varied from day to day with the temperature, but at times varied in the opposite direction. The operators believed that the cause was to be found in a combination of atmospheric conditions, including temperature, barometric pressure, and humidity, which undoubtedly would affect evaporation and cooling of water used either in towers or sprays. To what extent these conditions control the production or account for the variations in it, it is not possible to say.

Another plant that had been treating the same quantity of gas daily from the same wells for nearly three years suddenly increased its production from 5,200 to 5,900 gallons per day, averaged over a period of one month, no changes having been made in plant practice. The operators believed that the increase was due to some change in underground conditions that permitted the gas to be enriched. Exceedingly careful records have been kept at this plant, and these show that the daily production usually varied between 100 and 300 gallons. The sudden increase of 700 gallons daily over a period of one month has not been satisfactorily accounted for. The gravity of the condensate remained constant.

Usually the gravity of the total plant product increases in cold weather, owing to the separation of part of the heavy fractions in the gathering lines and to the condensation of lighter fractions in the water-cooled coils.

It appears that a satisfactory explanation of the causes of variation in production will not be arrived at until complete records of the variations in yield and gravity of the condensate, in the tem-

peratures and pressures used, and in atmospheric conditions are kept at different plants so these data can be studied with reference to each other and a comparison made of the results obtained at a number of plants in different fields and under different climatic conditions.

THE USE OF AMMONIA AS AN AUXILIARY COOLING AGENT.

DESCRIPTION OF PLANT IN FULLERTON FIELD.

A detailed description of plant 2, which is situated in the Fullerton, California, field follows: The casing-head gas being treated in the plant is brought from 20 oil wells through 3,500 feet of 2, 4, and 6 inch lines, 2-inch lead lines from the different wells being connected with the main gathering lines of 4 and 6 inches diameter.

The 6-inch main discharges the gas at the plant into a 6-foot by 10-foot steel receiver which also acts as a scrubber and accumulating tank for "line distillate," removing from the gas the crude oil, the condensate formed in the pipe lines, and dirt. The receiver is connected to a single-stage, 40-horsepower, noncondensing, direct-connected air compressor with 12 by 16 by 12 inch cylinders. The compressor is steam driven, rate 200 revolutions per minute, and takes steam at a pressure of 110 pounds. The compressor holds on the intake receiver a 10-inch vacuum that brings the gas through the pipe lines from the well, but is practically dissipated by friction and leakage, leaving the pressure at the wells at or near zero.

REFRIGERATION "STILLS."

Seven "stills," or coil heat interchangers are used in cooling the gas. (See fig. 11.) Each still consists of 1,260 feet of 1½-inch, extra heavy pipe coiled, with return bends, inside of a 12-inch tube 80 feet long laid at a slope of 1 inch in 10 feet, or 8 inches in all, to collect the condensate at one point, thence it is drained into storage or "make" tanks. The seven stills are parallel with one another, all draining in the same direction. Four of the stills, in which ammonia is used as the refrigerant, are insulated by about 1 foot of sawdust contained in a wooden housing about the tube; the other three coils, which treat the hot gas from the compressor, are not insulated and are exposed to the air, the hot gas flowing in the 12-inch tube and the cold gas in the 1½-inch coils.

The compressor discharges the gas at a pressure of 37 pounds and a temperature of 150° F. into stills 1 and 2, connected in parallel. These stills discharge into the outer tube of still 3, the gas from this still flowing through the outer tube of each of the other stills in succession. The dry, cold gas from still 7, in which the lowest temperature, approximately 10° F., and the final precipitation of condensate are obtained, is discharged into a pipe which returns it to the 1½-inch

coils of stills 1, 2, and 3. The gas is divided equally between the three coils of these stills, flowing through them in parallel, thence it discharges into a header connected with the field, or "dry-gas," lines carrying it back to the various leases.

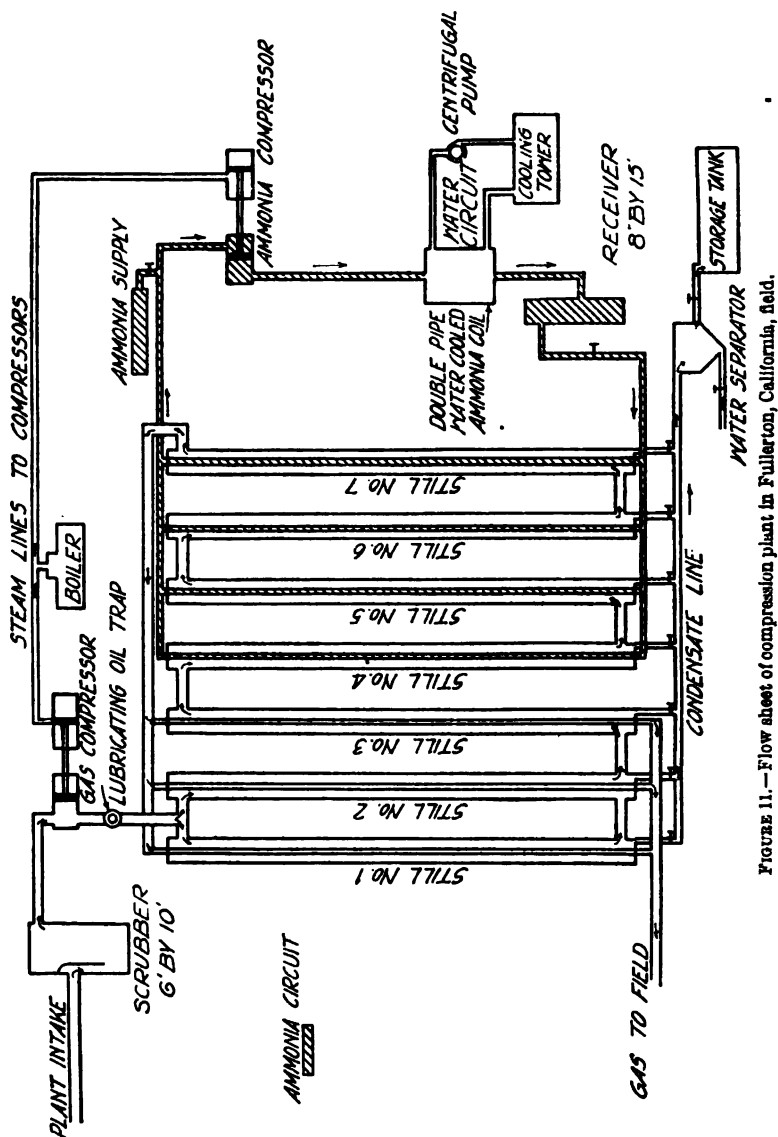


FIGURE 11. — Flow sheet of compression plant in Fullerton, California, field.

TEMPERATURES OBTAINED.

The following table gives the average temperature of the gas as it enters each of the stills; also the average gravity of the condensate discharged at the low end of each still. The gravity of the mixture of the entire product from all the stills averages 76° B.

Temperature of gas and gravity of condensate from each of the seven stills.

	Temperature of gas in each still, °F.	Gravity of condensate, °B.
Stills 1 and 2.....	150	62
Still 3.....	75	65
Still 4.....	50	73
Still 5.....	40	78
Still 6.....	30	85
Still 7.....	10	95

COOLING SURFACE.

In each still the 1,260 feet of 1½-inch pipe exposes a radiating surface of 412 square feet to the wet gas being refrigerated in the 12-inch tube, or 2,884 square feet in the entire set of seven stills. Of the 2,884 square feet of surface area, 1,648 feet are cooled by ammonia in stills 4, 5, 6, and 7, and 1,236 feet by cold, dry gas in stills 1, 2, and 3 that has passed through the entire set of refrigerating tubes, giving a total of 8.24 square feet of cooling surface for each 1,000 cubic feet of gas treated per day.

COLLECTING CONDENSATE.

The condensate from each still is drawn off continually from the bottom at the low end through a 1-inch pipe manifold to a cone-bottom settling tank in which the gasoline and the water separate by gravity, the water being drawn off at the bottom and the condensate flowing to the storage tanks. The 1-inch manifold and the bottom of each still are connected by a ½-inch gage glass through which the condensate precipitated in that still flows, allowing the operator to see at all times the flow of condensate before it is mixed with that of other stills. By this arrangement he can note, without stopping the plant, whether any discolored condensate is being discharged, or whether any one of the stills is not working properly.

AMMONIA CIRCUIT.

The ammonia used in refrigeration is compressed in the duplex compression cylinders of a 30-ton Stevens ice machine, direct-connected to a 125-horsepower, tandem-compound, steam-driven Corliss engine, with 10 by 20 by 12 inch cylinders taking steam at 110 pounds and operating at 80 revolutions per minute.

From the compression cylinders the ammonia gas is discharged at a pressure of 150 pounds to the inside 1½-inch pipe of a double-pipe water-cooled coil, the water circulating through the outside 2-inch pipe. This coil unit consists of three sets of double-pipe return-bend coils eight pipes high and 20 feet long. Water circulated by a centrifugal pump flows from the coils over a cooling tower, collects in the tower basin and is returned to the coils by the pump at a temperature somewhat below that of the atmosphere. From the inside coils the ammonia flows through a ½-inch pipe to a receiver or storage tank,

made of 8-inch casing 15 feet long, thence through an expansion valve to the four stills, connected in parallel as described above, at a pressure of 15 pounds and a temperature of 10° F. The ammonia is discharged from the four stills into a pipe manifold leading to the ammonia compressor, to be returned through the circuit. Ammonia lost in the pipes and stills by leaks and breaks is replaced from time to time from a steel bottle of compressed ammonia connected to the ammonia circuit as indicated in figure 11.

DESCRIPTION OF PLANT IN SANTA MARIA FIELD.

Plant 10 uses ammonia as an auxiliary cooling agent in addition to water cooling and expansion cooling. At this plant the gas passes through water-cooled coils after each of the three stages of compression, then through a coil cooled with brine refrigerated by ammonia from an ice-making machine, and then through double-pipe coils cooled by expanding gas.

The gas, after being cooled in the high-pressure (250 pounds) water-cooled coils, is led through a continuous coil of 4-inch pipe 450 feet in length, inclosed in a wooden tank or basin built with double walls and bottoms, 9 inches apart, the space between the walls being packed with sawdust. This coil is cooled with brine. The tank bottom has enough slope to drain the brine toward one end, whence it is pumped for recirculation through the unit. The brine (calcium-chloride solution) is brought in contact with the expanded ammonia by the use of coils in an iron tank, reducing the temperature of the brine to about 32° F. From this cooling tank the brine is circulated by a centrifugal pump to the gas-cooling coils, where it is discharged in such a way as to drip over the coil and collect at the low end of the basin, to be discharged again to the ammonia-cooled brine tank.

The gas discharged from the brine-cooled coils has a temperature of 32° to 34° F. The advantage claimed for this system is that the temperature produced by the brine cooling precipitates all the water vapor in the gas, thus preventing freezing of the double-pipe coils cooled by gas from the expansion engine. This is without doubt an advantage to be desired, but it could probably be obtained in this plant, as in other plants, by a more thorough use of expanded gas in coils of greater length and smaller diameter, or by using the expanded gas in two sets of coils and cooling the high-pressure gas in two stages, the first stage using the expanded gas from the second-stage coils. The first coils being partly warmed, would precipitate only water if the temperature were properly adjusted as is done in other plants described. The ammonia compressor and coils, also the brine circulating pumps, coils, and cooler could be abandoned, and only the extra set of expansion coils put in to replace them.

The brine and ammonia cooling installation is cumbersome, inefficient, and requires more time and care than the result warrants.

TREATMENT OF STILL VAPORS BY COMPRESSION AT A REFINERY IN NEW JERSEY.

VAPORS TREATED.

The gases treated in the compression plant, designated as plant 80 in the tables, at a refinery in New Jersey are those from all cooling coils in which the lighter fractions of crude oil and naphthas from both fire and steam stills are being condensed. Figure 12 shows (at the extreme left) the condenser box and coils from which the uncondensed gases and vapors treated by compression originate. From that point each unit and process is shown diagrammatically to the point at which the condensate and fixed gases are finally separated

COLLECTING VAPORS.

At the discharge pipe of the coils in water boxes a T connection is made, the condensate flowing down into pipes connected with the "tail house" and "look boxes"; the gases and vapors rise through a vertical pipe 6 feet high to the 8 or 10 inch collecting pipe called the gas main. The gas main is also connected with a 2-inch pipe to the condensate line at a point about 2 feet back of the look boxes, and with a 2-inch pipe leading from the top of the look boxes, both of which are used to relieve the pressure and collect vapors that have been carried past the first stage of separation, or have formed in the condensate flow lines. From the top of the gas main the gas is led through 12-inch pipe connections past a butterfly valve, which regulates the vacuum held on the discharge pipes of the condenser coils, to a vertical steel receiving tank 15 feet in diameter and 18 feet high. In this tank the gas is given a preliminary scrubbing with sea water, removing part of the sulphur compounds, some heavy oils which have been carried through the stills, and a small quantity of discolored condensate of approximately 53° B. gravity. The 12-inch pipes leading to the receiver are taken out at the top of the gas main, instead of the bottom, so as to trap back any condensate formed in the main and allow it to flow down the 6-foot risers and back into the lines from the coils to the tail house with the rest of the condensates produced. The vacuum held and regulated by the butterfly valve, previously referred to, on the gas mains and on the discharge of the condenser coils is between 0.25 and 1 inch of mercury (2 to 8 ounces below atmospheric pressure). There is no gage between the butterfly valve and the blower that produces the vacuum, so no record of the pressure between these points is available. However, the writer was informed that a test had been made that showed a vacuum of 22 inches of mercury.

the scrubbers. From the blower the gas passes through six 4 by 20 foot vertical iron scrubbers connected in series. The connections are so made that by opening or closing valves, the scrubbers may be operated as two sets, in parallel, of three tanks connected in series. This arrangement was tried and abandoned in favor of the series of six. In each scrubber the gas enters at a point about two feet above the bottom, passes a series of wood baffles over which warm wash water or caustic solution is falling, and discharges at the top. The gas then passes through the next scrubber in the series.

In the first four scrubbers warm sea water, flowing over baffles countercurrent to the gas, is used to remove impurities, consisting chiefly of sulphur and sulphur compounds, from the gas. In the fifth scrubber a solution of sodium hydroxide (lye) with a specific gravity of 12° B. is circulated over the wood baffles, in the same manner as the water in the other tanks, to remove any acids contained in the gas. These impurities may originate either in the crude oil or in the refining of the various distillates from which vapors are taken for treatment by compression. The lye used, as received at the refinery from the manufacturer, is a 35° to 40° solution. It is transferred to one of four iron storage tanks and diluted to the strength used in the scrubber. After being circulated until it will not neutralize the acid in the gases and becomes foul, it is wasted and fresh solution is put into circulation.

In the sixth scrubber warm salt water is used to remove any traces of caustic remaining in the gas. Caustic in the gas would react with the lubricating oils in the compressor cylinders, causing cutting of the cylinders or an excessive waste of oil.

The gas in passing through the six scrubbers is warmed 2° to 5° F. by the warm water used as the scrubbing medium. This warm salt water is the discharge from the Wheeler condenser used in connection with the low-pressure steam cylinder of the compressor. The fact that the water used is salt has nothing to do with the process. Sea water is used because it is the most available, the plant being situated on the Atlantic coast. No condensate is formed during the scrubbing. This would be anticipated because of the increased temperature of the gas due to the use of the warm condenser water. Both the water and the lye solution circulated through the scrubber tanks are handled by duplex pumps.

COMPRESSION AND COOLING.

From the sixth scrubber the gas is delivered at a pressure of 1 pound and a temperature of 70° F. to the low-pressure cylinder of the compressor, which discharges it at a pressure of 43 pounds and a temperature varying between 200° and 250° F.

Gas discharged from the low-pressure cylinder to the intermediate cooling coils is passed through a Bundy oil separator, which removes

lubricating oils carried over with hot gas and gasoline vapor from the compressor cylinder. The coils used are the water-cooled type submerged in a box, typical of refinery construction. The gas is divided in a header into six sets of return-bend coils of 3-inch pipe 10 feet long and six pipes high, totaling 360 feet of 3-inch pipe, exposing a radiating surface of 283 square feet, or 0.189 square foot per 1,000 cubic feet of gas treated per day. This is approximately one-third of the radiating area usually found in compression plants for the same service.

At the bottom of the coil the gas is again collected through a header and discharged into an accumulator tank in which 750 to 1,000 gallons of condensate is collected each day. The condensate varies in gravity according to weather conditions, averaging 68° B. in summer and 72° B. in winter. The product collected in this accumulator tank is forced into storage tanks by the working gas pressure as often as necessary and blended with the rest of the condensate. The gas leaving the coils has a temperature of 70° to 90° F., the high temperature probably being due to the small cooling area used at this plant.

At this temperature and pressure (70° to 90° F. and 43 pounds) the gas enters the high-pressure cylinder and is discharged at a pressure of 160 pounds and a temperature between 170° and 250° F. The gas is again led through a Bundy trap to separate lubricating oils, as previously described, and then to the high-pressure cooling coils, which are the same size and length as the intermediate coils used to cool the low-pressure gas, except that two sets are used in series in place of one, having twice the cooling area. The average temperature to which the gas is reduced in these coils is 70° F. and is the lowest temperature used in the treatment.

After the condensate and gas have been separated in the high-pressure accumulator tank the pressure is reduced through a valve to one-half pound and the gas discharged to a gas receiver in which gas is stored and used for fuel under boilers, stills, etc., in the plant.

BLENDING.

Blending at this plant is all done under a pressure of 160 pounds in the high-pressure accumulator tank, as follows:

Naphtha with a gravity of 53° B. is pumped through 1½-inch pipe in coil boxes and cooled to 70° F., then into the top of the high-pressure accumulator tank, and sprayed through the rising gas at a rate which gives a mixture containing approximately four parts of naphtha to one of condensate, or about 80 per cent naphtha. At regular intervals the mixture is drawn off into another tank containing the low-stage condensate, and the resulting mixture sampled and tested for gravity. If the gravity is found to be too high, more

naphtha is pumped into the accumulator tank, which lowers the gravity of the next batch drawn off into the storage tank and of all the blend in storage, or if the gravity was too low the proportion of naphtha pumped into the accumulator is cut down. From 4,000 to 5,000 gallons of raw condensate is made daily in the high-pressure coils, and this with the condensate from the low-pressure coils makes a total production of 5,000 to 6,000 gallons per day from treated gases and vapors. The raw condensate from the high-pressure coils has a gravity varying between 76° and 93° B. After blending with naphtha, the product has a gravity of 58° to 61° B.

The level of the mixture in the high-pressure accumulator tank is at all times kept above the discharge pipe from the coils, thus forcing all the gas to pass through the blended product. It is claimed that this method adds 250 to 500 gallons daily to the net production. Inasmuch as the blend is four parts naphtha to one of condensate, and the temperature of the gas in the coils is such as to leave part of the comparatively heavy gasoline fractions uncondensed, absorption of liquids from the gases or absorption of the gases themselves may reasonably take place. The general practice throughout the United States, however, is to remove the condensate from contact with the gas as soon as possible.

QUANTITY OF GAS TREATED, AND PRODUCTION.

The volume of gas treated is computed from the compressor displacement, and the record of engine revolutions with 5 per cent deducted for slippage. On this basis the gas passing through the plant varies between 1½ and 2 million cubic feet per day, and shows an average production of 3.09 gallons of condensate per 1,000 cubic feet of gas treated.

COMPRESSOR.

All gas compressed passes through a 2-stage Corliss-valve compressor, direct connected to a combination cross-compound, condensing Corliss engine; steam cylinders are 16 and 32 inches in diameter with 30-inch stroke. The compressor speed varies greatly, being dependent upon the amount of gas coming from the stills, which in turn depends upon the stage of distillation of oil at which the various stills are working.

SPECIAL FEATURES OF PLANT.

The noteworthy features of this compression plant are the scrubbers for removing sulphur and acid, the Root blower as a booster unit, the passing of gas through the condensate to bring about greater production of gasoline condensate by absorption, and the large amount of naphtha used in the blended product.

MACHINES USED IN COMPRESSION PLANTS.

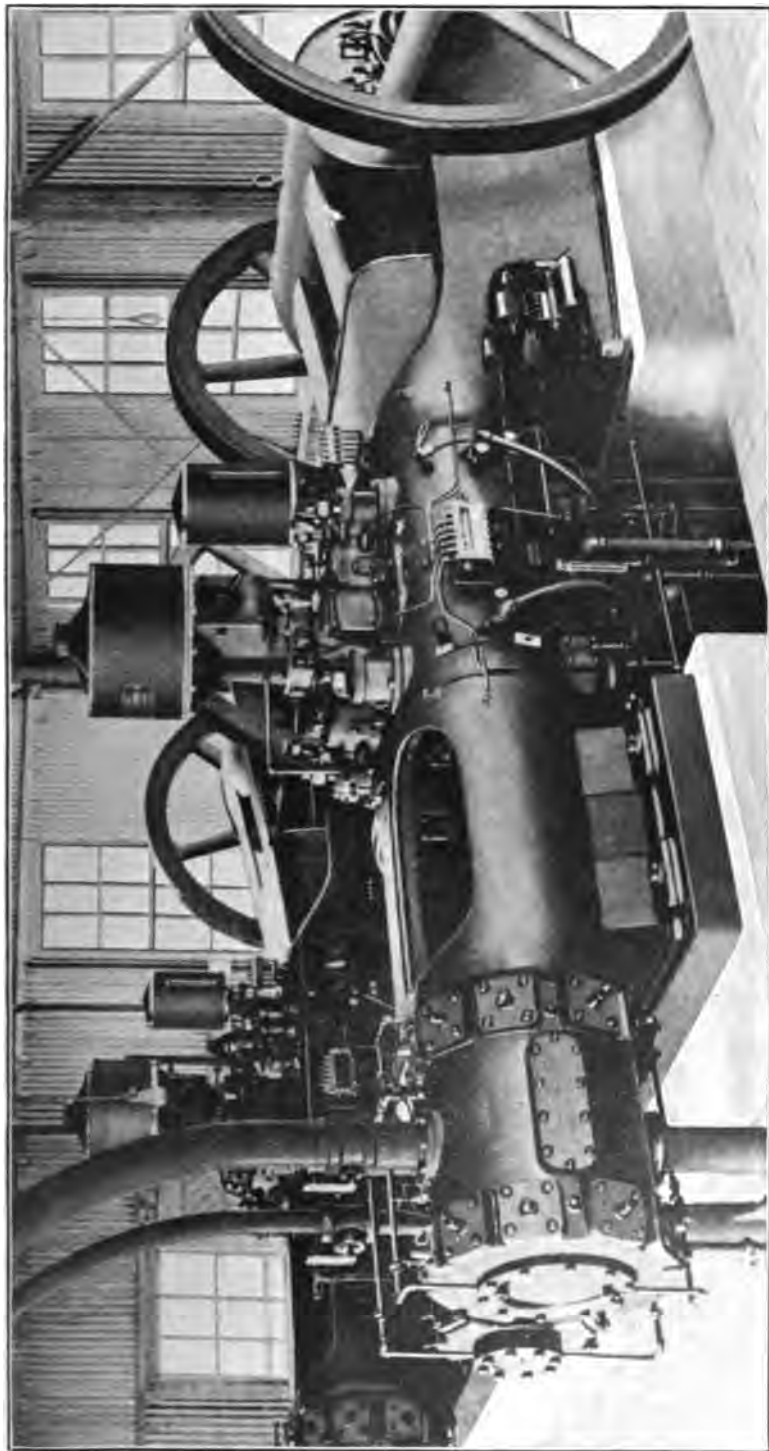
Table 7 shows the total rated horsepower used in gas compression in each of the plants listed, and the number of cubic feet of gas compressed per day by one rated horsepower.

TABLE 7.—*Rated horsepower and capacity of various plants.*

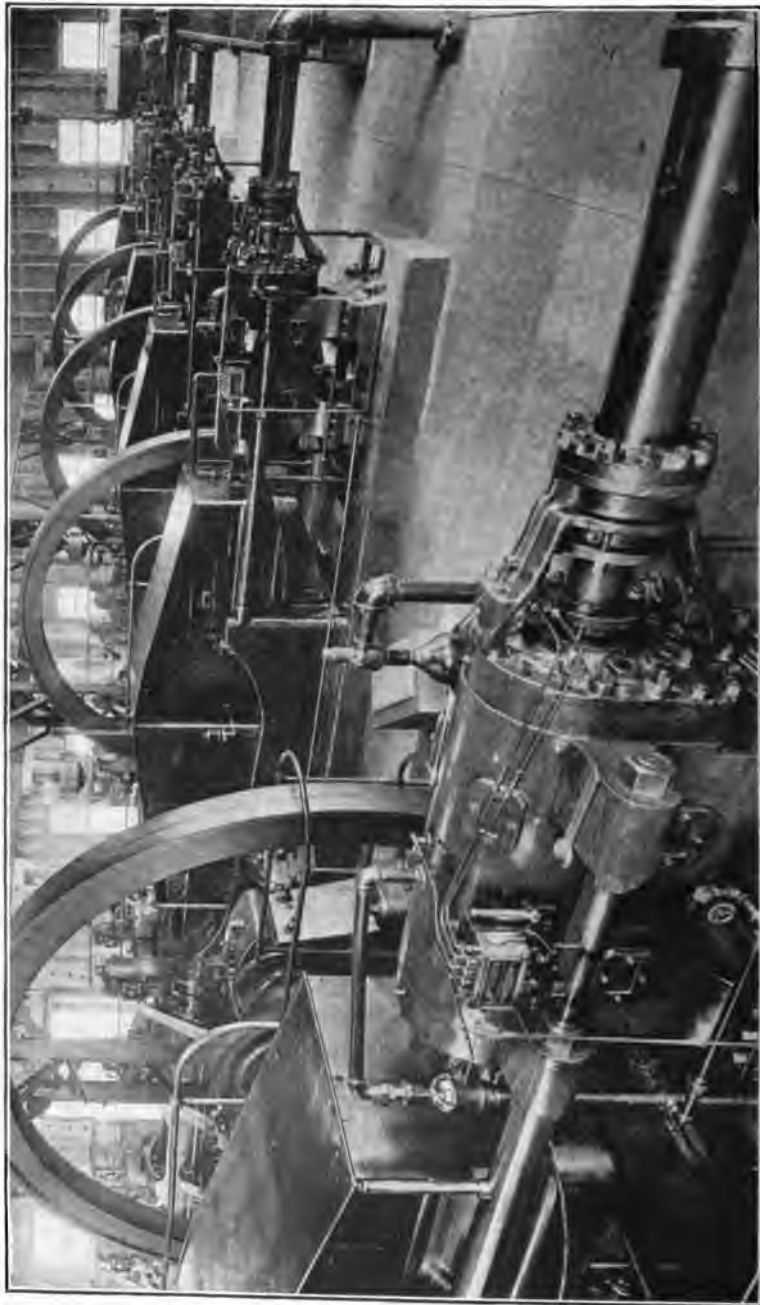
Plant No.	Total rated horsepower.	Cubic feet of gas compressed daily per horsepower.	Plant No.	Total rated horsepower.	Cubic feet of gas compressed daily per horsepower.
3	110	4,100	34	300	2,500
4	300	4,000	35	100	3,500
7	1,800	4,160	36	480	2,400
9	320	3,200	37	170	2,300
11	480	3,100	38	300	2,500
12	240	2,917	50	320	3,700
14	330	3,030	51	110	2,170
16	100	2,500	52	100	2,500
17	300	2,500	54	900	2,500
19	575	3,080	55	300	4,170
20	495	3,030	56	210	1,800
21	300	3,300	57	350	2,400
22	180	3,300	76	300	6,750
30	210	3,500	77	50	8,000
32	695	4,300	78	50	900
33	560	3,200	79	105	2,380

The quantity of gas compressed or treated per horsepower in the plants visited varies between wide limits. The chief causes for this variation are that the intake pressures vary widely and also the final pressures vary between 75 and 300 pounds. Many other factors in plant operation also affect the results, among which are the actual efficiency of the engine and compressor used, the temperature of the gas at different stages, the sizes and length of pipes through which the gas is forced, and the power used in driving pumps and line shafts, which in many plants is taken from the engine that drives the compressor. In plants where expansion engines are used to compress gas no attempt has been made to estimate or add the power delivered by such units.

The table shows that the average plant visited by the writer has 340 horsepower and treats approximately 3,250 cubic feet of gas per day for each rated horsepower installed. The conditions under which many of these plants are operating may be found in Tables 3, 4, and 5. Among operators who design plants and use the rated horsepower, rated compressor capacity, and atmospheric pressure at the intake, with a discharge pressure of 250 pounds, 4,000 cubic feet of gas daily per horsepower is used for preliminary estimates of the requirements of a plant to treat a given volume of gas. The final estimates necessarily must include all the items mentioned above, and also take into consideration all of the special conditions under which the plant in question is to operate.



TWO-STAGE DIRECT-CONNECTED COMPRESSOR AND GAS ENGINE.



FIVE 485 BRAKE-HORSEPOWER GAS ENGINES, FRONT-CONNECTED TO 15 $\frac{1}{2}$ -INCH COMPRESSOR CYLINDERS, AT A WEST VIRGINIA PLANT THAT COMPRESSES NATURAL GAS TO A PRESSURE OF 275 POUNDS GAGE DISCHARGE. IS FIELD END OF A 20-INCH PIPE LINE.

ENGINES AND POWER.

Machines of practically all well-known manufacturers making gas engines or compressors have found their way into compression plants. One company which has done much pioneer work in both the laboratory and the field, tending to develop the natural-gas gasoline industry, has placed its engines, compressors, or both, in many plants in every field visited by the writer.

Plates XIII and XIV show engine installations at two compression plants.

Table 8 gives data on the motive power, types, and sizes of engines and compressors, method of connection, and the number of compressor units used in the plants visited.

In plant 24, electric motors have been installed to drive the compressors. Originally gas engines were used, but the gas being treated became so rich in gasoline content that not enough gas was discharged after treatment to operate the engines. The motors are set on the blocks that were used for engine beds and are in the same room with the compressors. Because of the possibility that gas may escape about the compressors and be ignited by sparks from the motors, this arrangement would appear to be dangerous practice, but thus far, owing probably to especial precautions in ventilation, no fires or explosions have occurred.

Plants 19, 20, and 21, all built in 1916, use vertical 4-cylinder, 4-cycle gas engines belted to the compressor units. While this type of machine has many moving parts which may get out of order, and a long crank shaft with its bearing to watch and tighten, it is giving complete satisfaction and is claimed to economize fuel and give an overload capacity of 25 per cent.

The 450-horsepower, direct-connected, 4-cycle, double-acting machines, although recognized as standard installations in gas-pumping plants throughout the country, have a number of disadvantages as units for the treatment of gas for gasoline extraction. A machine of this size should be installed only in plants that draw gas from areas large enough to insure a supply for a long term of years, and are of such capacity that one of these large units represents only a small fraction (10 to 20 per cent) of the total plant capacity, otherwise shutting down one unit unbalances the entire plant and materially reduces the output.

TABLE 8.—Data on engines and compressors

Plant No.	Engine.				
	Driven by—	Rated horse-power.	Type. ^a	Number of compressors used. ^b	Drive.
1.	Steam.....		Cross compound.....	2.	Direct.....
2.	do.....	40	Straight line.....	2.	do.....
3.	Gas.....	110	Straight-line, tandem, 2-cycle	1.	Belted.....
4.	do.....	150	Twin cylinder.....	2.	do.....
	Steam.....		Cross compound condensing.	1.	Direct.....
	do.....		Tandem compound condensing.	1.	do.....
6.	do.....		do.....	1.	do.....
	High-pressure gas. ^c		Cross compound, 12 by 25 by 30.	1.	do.....
	Gas.....	450	Tandem, 4-cycle, double-acting.	2 low.....	do.....
7.	do.....	450	do.....	2 high.....	do.....
	High-pressure gas. ^c		Cross compound.....	1.	do.....
8.	Gas.....				do.....
9.	do.....	80	Type VII, 16 by 20.....	2 high and 2 low.....	do.....
	High-pressure gas. ^c		Cross compound, 12 by 18 by 10.	1.	do.....
	Steam.....		Cross compound, 12 by 20 by 12.	2.	do.....
10.	do.....		Simple, 12 by 12.....	1.	do.....
	High-pressure gas. ^c		do.....	1.	do.....
11.	Gas.....	80	Type VII, 16 by 20.....	6, 3 high and 3 low.....	do.....
	High-pressure gas. ^c	50	Simple, 12 by 12.....	1.	do.....
12.	Gas.....	80	Type VII, 16 by 20.....	4, 2 high and 2 low.....	do.....
	High-pressure gas. ^c		Drilling engine, 9 by 12.....	1.	Belted to pump.....
14.	Gas.....	110	Twin.....	3.	Belted.....
	High-pressure gas. ^c		Simple, 10 by 12.....	1.	Direct.....
16.	Gas.....	50	Type VII, 13½ by 18.....	2, 1 high and 1 low.....	do.....
	High-pressure gas. ^c		Simple, 10 by 12.....		do.....
17.	Gas.....	150	Twin cylinder.....	2.	Belted.....
	High-pressure gas. ^c		Simple, 12 by 11.....	1.	Direct.....
	Gas.....	175	Vertical, 4-cylinder, 4-cycle.....	3.	Belted.....
19.	High-pressure gas. ^c		Duplex pump, 10 by 12 by 6.....	2.	
20.	Gas.....	165	Vertical, 4-cylinder, 4-cycle.....	4.	do.....
21.	do.....	150	do.....	4.	do.....
22.	{ do.....	Two 40	Type VII.....	4, 2 high and 2 low.....	Direct.....
23.	{ do.....	Two 50	Single cylinder.....	1.	Belted.....
24.	Electricity.....	35	Motor.....	2.	do.....
25.	Gas.....	50	Type VII.....	2.	Direct.....
27.	do.....	85-100	Horizontal.....	2.	Belted.....
28.	do.....	100	do.....	1.	do.....
29.	do.....	50	Single cylinder.....	1.	do.....
30.	do.....	70	do.....	3.	do.....
31.	do.....	110	Twin.....	2.	do.....
32.	{ do.....	165	do.....	9, 4 running.....	do.....
	High-pressure gas. ^c		Cross compound, 12 by 19 by 18.	1.	Direct.....
33.	{ Gas.....	Eight 50	Type VII.....	10, 5 high and 5 low.....	do.....
	{ High-pressure gas. ^c	Two 80	Cross compound, 8½ by 12 by 12.	1.	do.....
34.	Gas.....	50	Type VII, 13½ by 18.....	6, 3 high and 3 low.....	do.....
37.	{ do.....	70	Single cylinder.....	1.	Belted.....
	{ do.....	50	Type VII, 13½ by 18.....	2, 1 high and 1 low.....	Direct.....
38.	do.....	50	Single-cylinder, 13½ by 18.....	6, 3 high and 3 low.....	do.....
50.	do.....	80	Type VIII, 16 by 20.....	1.	do.....
51.	do.....	110	Twin-cylinder.....	1.	Belted.....
52.	do.....	50	Type VII, 13½ by 18.....	2, 1 high and 1 low.....	Direct.....

^a Figures show size of cylinder in inches.^b In this column "low" refers to low pressure, "high" to high pressure.

used at various compression plants.

Compressor.			Rated speed (r. p. m.).	Remarks.
Description.	Size of cylinder (inches).			
	Low-pressure cylinder.	High-pressure cylinder.		
2-stage.....	21 by 24.....	9½ by 24.....	62	Two 300-hp. Sterling and four 150-hp. Erie boilers used.
Single-stage.....	12 by 16.....		200	
2-stage.....	13½ by 14.....	6½ by 14.....	160	
do.....	16 by 16.....	8 by 16.....	180	
2-stage.....	20 by 24.....	9½ by 24.....	125	
do.....	15 by 24.....	7 by 24.....	145	
do.....	15 by 16.....	7½ by 16.....	115	
do.....	24 by 30.....	12 by 30.....	38	
Single-stage.....	31 by 36.....		125	
do.....		15½ by 36.....	125	
Single-stage (duplex). ..	Two 22 by 16.....			
Single-stage.....	14 by 20.....	7½ by 20.....	200	
2-stage.....	16 by 10.....	8 by 10.....	100	
do.....	20 by 12.....	12 by 12.....		Used as low and intermediate at 12 pounds and 80 pounds. Used as high at 250 pounds. Used as air compressor.
Single-stage.....	14 by 12.....	8 by 12.....	Varied.	
do.....	Three 14 by 20.....	7½ by 20.....	180	Pumps dry gas at 60 pounds.
do.....	14 by 12.....		145	
do.....	Two 7 by 20... ..	Two 14 by 20... ..	190	
2-stage.....	13 by 14.....	6½ by 14.....	170	
2-stage, tandem.....	8 by 12.....	4 by 12.....	200	
Single-stage.....	11 by 18.....	5½ by 18.....	180	
do.....	12 by 12.....			
2-stage.....	16 by 16.....	8 by 16.....	150	
Single-stage.....	20 by 11.....		78	
2-stage.....	16 by 16.....	8 by 16.....	158	Engine speed, 300 revolutions per minute. Water discharge throttled to 100 pounds. Drilling engine, 10 by 12-inch cylinder, used as expansion engine. Engine speed, 275 revolutions per minute. Drilling engine, 10½ by 12-inch cylinders, used as expansion engine.
do.....	16 by 16.....	8 by 16.....		
do.....	16 by 16.....	8 by 16.....	150	
Single-stage.....	11 by 15.....	7 by 15.....	115	
2-stage.....	14 by 10.....	7½ by 10.....	120	
Single-stage (to 150 pounds). ..			180	
2-stage.....	12½ by 14.....	6 by 14.....	180	
do.....	12½ by 14.....	6 by 14.....	180	
do.....	10½ by 12.....	5½ by 12.....	180	
do.....	14 by 14.....	7 by 14.....	180	
do.....	16 by 16.....	8 by 16.....	180	
Single-stage (duplex). ..	18 by 18.....		180	
Single stage.....	Eight 12 by 18.....	Eight 6½ by 18.....		Feather valves on compressor.
Single-stage (duplex). ..	Two 14 by 20.....	Two 7 by 20.....	200	
do.....	14 by 12.....			
2-stage.....	12 by 18.....	9 by 18.....	100	
Single-stage.....	12 by 12.....	6 by 12.....	180	
do.....	10 by 18.....	5½ by 18.....	180	
do.....	12 by 18.....	6 by 18.....	180	
do.....	14 by 20.....	7 by 20.....	180	
2-stage.....	13 by 14.....	6 by 14.....	160	
Single-stage.....	12 by 18.....	6 by 18.....	180	

* Expansion engine operated by expanding high-pressure, treated gas through it.

TABLE 8.—Data on engines and compressors

Plant No.	Engine.				
	Driven by—	Rated horse-power.	Type.	Number of compressors used.	Drive.
53.....	{ Gas.....	{ ^a 80	Type VII.....	2, 1 high and 1 low....	Direct.....
	{ do.....	{ ^b 50	Type VIII.....	do.....	do.....
	{ do.....	80	Twin.....	1.....	Belted.....
	{ do.....	150	2-cylinder, tandem.....	1.....	do.....
54.....	{ do.....	50	Type VII.....	6, 3 high and 3 low....	Direct.....
	{ do.....	150	Twin-cylinder.....	2.....	Belted.....
55.....	{ do.....	150	do.....	2.....	do.....
	{ do.....	50	Type VIII.....	6, 3 high and 3 low....	Direct.....
56.....	do.....	70	Single-cylinder.....	3.....	Belted.....
57.....	do.....	70	do.....	5.....	do.....
58.....	do.....	50	Type VIII.....	do.....	Direct.....
59.....	do.....	50	do.....	do.....	do.....
60.....	do.....	50	do.....	do.....	{ do.....
61.....	do.....	do.....	do.....	do.....	{ Belted.....
62.....	do.....	50	Type VIII.....	do.....	do.....
76.....	{ do.....	50	Type VIII, 13½ by 18.....	4.....	Direct.....
	{ High-pres- sure gas.c	do.....	Duplex, 13½ by 18.....	1.....	do.....
77.....	{ Gas.....	50	Type VIII.....	do.....	do.....
	{ High-pres- sure gas.c	do.....	do.....	do.....	do.....
78.....	{ Gas.....	50	do.....	1.....	Direct.....
	{ do.....	35	Pump, 10 by 6 by 12.....	2.....	do.....
79.....	do.....	do.....	Single cylinder.....	2.....	Belted.....
80.....	Steam.....	do.....	Cross compound, Corliss, 16 by 32 by 30.....	1.....	Direct.....

^a Low pressure.^b High pressure.

used at various compression plants—Continued.

Compressor.			Rated speed (r. p. m.).	Remarks.
Description.	Size of cylinder (inches).			
	Low-pressure cylinder.	High-pressure cylinder.		
Single-stage				
do				
2-stage	15½ by 16.	7½ by 16.		
do	14 by 14.	6½ by 14.		
Single-stage	11 by 18.	6 by 18.	180	
2-stage	16 by 16.	7 by 16.	180	
do	16 by 16.	7 by 16.	180	
Single-stage	11 by 18.	5½ by 18.	180	
2-stage	12 by 12.	6 by 12.	180	
do	12 by 12.	6 by 12.	180	
.....				
.....				
Single-stage	11½ by 18.			
2-stage				Compresses air in two stages to 85 pounds.
Single-stage	11½ by 18.			
do				
.....	11½ by 18.			
2-stage	12 by 12.	6 by 12.		Feather valves on compressor.
do	25 by 30.	13 by 30.		Corliss valves on compressor.

* Expansion engine operated by expanding high-pressure, treated gas through it.

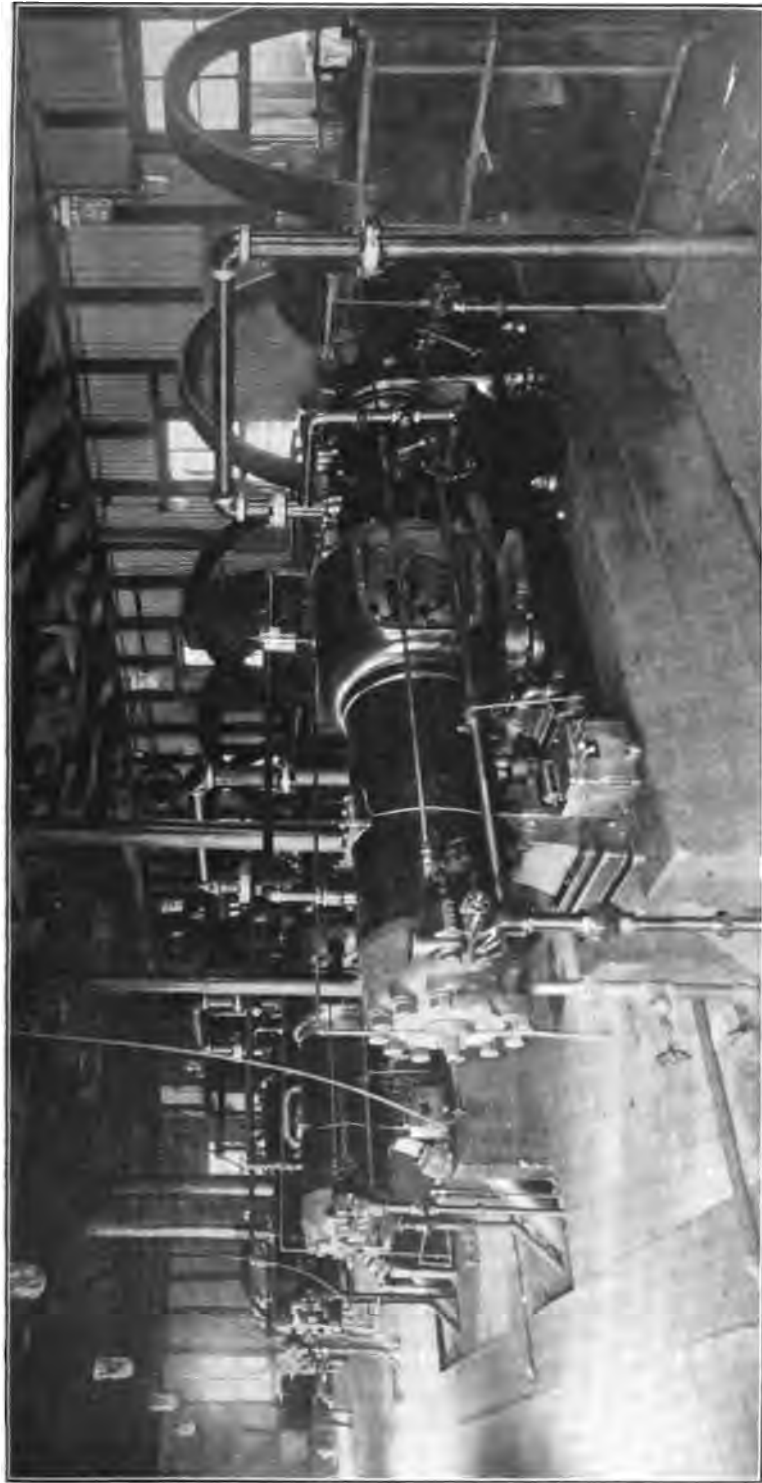
In the casing-head gasoline industry the gas supply from a given area is expected to decline and the pressure necessary to precipitate the condensable product usually becomes lower as the gas becomes richer. If all the power capacity thus relieved from duty is not used to increase the vacuums held on the lines, it is obvious that to remove one unit to another locality would be desirable. If one unit represented half the total plant capacity, it would be necessary to wait until the available supply of gas had been reduced 50 per cent before that unit could be transferred, but if that unit represented only 20 per cent of the plant capacity it could be removed when the gas had fallen off that amount; also it would be a much simpler matter to find a location or sale for a machine of one-half million feet capacity than for a machine requiring from three to four million feet per day. Another argument for machines of smaller capacity is the size and weight of parts which under the usual transportation conditions in oil fields is a serious problem in plant construction.

Both direct-connected (Pl. XV) and belted units are being installed in compression plants using the usual smaller units. However, many operators of wide experience prefer the belted drive, claiming that it simplifies repairs and renewals and gives a wider choice both of engine and compressor. The belted type requires larger buildings, because of the necessary distance between pulley centers, which the writer found to vary between 22 feet for vertical engines to 34 feet for horizontal types. In some plants two buildings are used, one containing the engines and the other the compressors, with belt galleries connecting the two. This is done as a precaution against fire and explosions.

TYPES OF MACHINES USED AS EXPANSION UNITS.

All expansion engines used in compression plants are machines originally designed to use steam that have been slightly modified to overcome the effects of the low temperatures from the use of compressed gas in the steam or power cylinders. Among the machines found in use are direct driven reciprocating pumps, drilling engines, and simple and cross compound compressors. Of these the reciprocating pump is the least to be recommended. The valves tend to freeze and stick, owing to the general design and slow action, and its efficiency as a power unit and its capacity are low.

Of the machines used as expansion engines the converted steam engine of the single or double stage (simple or compound) type is more often found than any other make. On these engines, remodeled for the expansion of high-pressure gas, the valve mechanism has been designed especially and made stronger for this particular service. The exceedingly low temperatures developed in the cylinders tend to freeze the valves, causing excessive strains on the valve stems and rods, and also make lubrication difficult.



DIRECT-CONNECTED COMPRESSOR AND GAS ENGINES.

Glycerin has universally been found to be the best lubricant for expansion cylinders and was in general use until the recent rise in price, and the difficulty of obtaining a reliable supply forced plant operators to try other oils, some of which have been found usable but not as satisfactory as glycerin. As the supply of that lubricant increases it will indoubtably again be generally used.

Other types of steam compressors and, in several plants, drilling engines belted to plunger or centrifugal pumps, are in use as expansion units, and, according to the operators using them, are giving satisfaction in regard to both capacity and temperatures. The loads of the drilling engines were varied to meet the conditions of pressure and volume of gas by throttling the flow of water from the pump, thus giving the required resistance for the development of power at normal engine speeds. The valve mechanism of the drilling engines was rebuilt and strengthened to meet these conditions.

At one plant where a drilling engine gave much trouble from freezing the piston rings were removed and the piston dressed to a square edge. The piston in this condition shaved the ice from the cylinder walls, thus preventing sticking or freezing. No lubricant was used in the cylinder, the film of ice or frost on its walls acting as a lubricant, and leakage past the piston after the film had formed was very small.

At plant 3 an engine of this type that has a 9 by 12 inch cylinder, making 90 revolutions per minute with one-fourth cut-off, takes gas at 130 pounds, gas at 250 pounds being throttled to that pressure, and exhausts it at 10 pounds. The engine is belted to a three-plunger pump throttled to a back pressure of 160 pounds in order to increase the load on the engine. The water pumped is used in the cooling towers and in the engine and the compressor jackets.

At plant 16 the expansion engine, accumulator tanks, and storage tanks are all housed in a double-wall insulated building. The advantage claimed is that the condensate is held at a low temperature in the storage and other tanks, thus preventing loss by evaporation through heating before blending. The condensate precipitated in the coils is kept from sudden and extreme rises in temperature, as would be the case if the storage tanks were not cooled or shaded. The temperature in the building and of the condensate in storage is thus held between 34° and 50° F.

Expanding the compressed, treated gas in engines to develop power for driving compressors or pumps is not an economy, because of the small amount of actual power delivered, the care necessary, and the cost of upkeep and lubrication, such engines being used only because the cooling effect the expanded gas has in the treatment of high-pressure gas in double-pipe coils or heat interchangers.

In simple expansion engines such as pumps, drilling engines, or single-expansion compressors the compressed gas before entering the

power cylinder is often reduced by a throttle valve from the maximum pressure used in the plant to pressures varying between 120 and 160 pounds, depending upon the design and size of the machine and the pressures for which it was built and under which it gives the best results.

Cross-compound machines usually take the gas at the maximum plant pressure and reduce the pressure in the first expansion to 30 to 60 pounds (see Table 5) and in the second stage to between 5 and 15 pounds. Generally the gas goes from the first-stage expansion cylinder to a drip or small accumulator tank, then directly to the low-stage cylinder. In plant 6 (see Table 4, p. 41), the gas from the first-stage expansion is led through a set of double-pipe coils and heated before being put through the second stage of expansion. By this system both the power developed in the expansion engine and the cooling efficiency of the expanded gas are increased, also the danger of freezing in the expansion cylinders is less.

COMPRESSORS.

Direct-connected compressor units are usually of the single-cylinder engine, single-stage type, although some plants in the eastern and Mid-Continent fields are using direct-connected, twin 2-stage machines. Of the direct-connected compressors the single-stage unit, found more often than any other make, as shown in Tables 8 and 9, has the engine and compressor cylinders opposed, or on the opposite sides of the fly wheels.

The sizes and types of compressors used in the various plants visited by the writer are shown in Table 8 (p. 74), which conveys an idea of the great variety of compressors and drives used in such plants.

AUXILIARY MACHINES.

In all natural-gas gasoline plants small engines are needed to circulate water in jackets, towers, and ponds, to produce electricity for light and pump air to start gas engines. The number, situation, and type and size of these units are generally matters of preference with each plant operator and no standard has been followed.

WATER-CIRCULATING PUMPS.

Many of the newest plants have centrifugal pumps belted to a pulley on the engine, which circulate the water used in both engine and compressor cylinder jackets; other plants use a line shaft with pulleys belted to the engines and to the centrifugal pumps. In many fields the water is heavily charged with mineral salts, and the water used in the cylinder jackets is often distilled or condensed and kept separate from that used in cooling the gas coils. This is often done by cooling the condensed water in a separate tower or by passing it through coils

in the main gas-cooling tower that are cooled with spray in the same way that the gas coils are cooled. In the latter method the condensed water is pumped from the coils in the tower through the jackets and back to the coils in a closed circuit. Where a separate tower is used the condensed warm water is sprayed in the tower, collected in the basin beneath the tower, and circulated by pumps through the jackets, and returned to the top of the tower to repeat the cooling and aeration. In this method the loss of condensed water in the tower is so great that it is not to be recommended unless the condensed water is easily made or obtainable from a boiler plant. The cooling effect gained is better, and the installation less expensive, if the cost of condensed water is not an object.

For circulating the water used to cool the gas, usually centrifugal or plunger pumps driven by small (5 to 10 horsepower) gas engines, or belted to a line shaft, are installed.

AIR PUMPS.

Air pumps for compressing air in receivers to start the main gas engines are generally single-cylinder pumps or compressors belted to gas engines, previously mentioned, or, in some plants, to a line shaft driven by a small gas engine. The receivers are built to stand pressures up to 250 pounds. When the desired pressure is built up in the receiver, the air pump is shut down until the pressure is relieved by use in starting the main gas engines, or by leakage, when the pump is again operated until the required pressure is obtained.

LIGHTING EQUIPMENT.

The necessity of using strong, reliable inclosed lights around compression machinery and the danger of open flames in plants treating natural gas at high pressures has forced operators to install small electric generators as part of the equipment of gasoline plants.

The electric plant is always housed in a building separate from the compressors, and usually in the building with the air and water pumps. The unit consists of a gas engine belted to a dynamo of a size and capacity suitable to the lighting needs of the plant, and is operated only at night. In large gas-pumping stations storage batteries are used both for light and for engine ignition, in place of direct connections.

GASOLINE PUMPS.

Plants situated some distance from the shipping point or blending station often require pumps to force the gasoline through the pipe lines to such stations.

The product is often blown from one tank to another by turning high-pressure gas into the tank containing the product, but where the

distance is great this method is not always satisfactory, and pumps of either rotary or reciprocating type are installed, usually being operated by belt connections to small gas engines. It is claimed for the rotary pumps that condensate losses are smaller than from the use of other types, because the agitation is less and the flow more steady and quiet through the pump and into the lines.

BLENDING AND SHIPPING THE CONDENSATE.

Although blending of compression-plant condensates is not done primarily for the purpose of making a product that will meet the specifications required by transportation laws governing the shipment of gasoline by rail or water, blending and shipping have become such important functions, one of the other, that a separate discussion of the blending process and the transportation of the blended stock is not desirable.

REASONS FOR BLENDING.

When the condensate produced by compression is allowed to weather unblended until its vapor tension is reduced to less than 10 pounds at 100° F. and its temperature rises to atmospheric, losses ranging up to 75 per cent of the total product often result, whereas if the condensate is mixed (blended) with heavier straight still-run refinery distillates the losses from weathering are reduced, usually to one-half that amount, and often more. This fact has led condensate producers to take advantage of blending to increase the volume of the product actually marketed, thus increasing their profits and also the supply of marketable motor fuels so desirable under present conditions. The development in gasoline motors up to the present time has not reached a stage that would make the heavier still distillates, such as are used for blending, a convenient or economical fuel if used as made, because of the difficulty in starting the motor with such fuel and its tendency to deposit carbon in the cylinders and on the pistons from incomplete combustion, causing "engine trouble."

Condensate produced by compression is also an undesirable fuel for gasoline engines. It is exceedingly volatile, which causes losses in handling, is dangerous because fumes are easily formed, and gives less power as compared with equal volumes of heavier distillates, a larger number of gallons being required to develop the same power. It gives a quick, sharp explosion in a motor cylinder, but seems to lack "push" after the explosion has taken place. In the above qualities it is in no way different from still-run products of similar gravity and similar end points, both products needing additions of less volatile, heavier, and more powerful fractions in order to form the most convenient and economical motor fuel.

The lighter fractions of petroleum distillates, as compared with the heavier products, have a lower calorific value per gallon but a

higher calorific value per pound. As all products of petroleum are sold in the United States by volume or liquid measure, the standards for comparison must be made on the heat units per volume and not per weight.

As previously stated, another important factor in blending is transportation. The Interstate Commerce Commission rules controlling shipments of petroleum products and liquefied natural gas allow transportation of petroleum distillates having vapor tensions of less than 10 pounds per square inch in standard tank cars, and products with vapor tensions of 15 pounds per square inch in specially built insulated tanks. As many plants produce condensate that has a vapor tension of 30 or more pounds as it comes from the accumulator tanks, blending and weathering are both resorted to by most manufacturers in order to bring the product within shipping rules, to increase the quantity, and to improve the quality of the product.

MARKETING UNBLENDED CONDENSATE.

A small quantity of natural-gas gasoline finds its way to consumers unblended. It is sold as "gas-machine gasoline," a product with a gravity of 80° to 86° B., used to make gas for certain domestic, commercial, and chemical purposes, and as "export gasoline," a product with a gravity of 74° to 80° B. and of 4 to 6 pounds vapor tension, which is usually sold in containers to foreign trade.

The great bulk of condensate, however, is blended in one way or another before it reaches the consumer, but not always completely blended at the plant where it is made, or by the producer.

In many eastern fields the condensate is held in storage under pressure until a given quantity is ready for shipment, when it is forced by gas pressure or pumps through small (2-inch) pipe lines to the tanks of firms making a specialty of blending and marketing motor fuels, and having blending stations centrally situated among the compression plants from which they receive condensate. Products having a gravity as high as 84° B. are shipped in this way to blending companies.

SHIPPING BY AUTO TRUCK.

Another method of shipment, used mostly in California and northern Pennsylvania, is by tanks mounted on auto truck. Two plants shipping condensate of 80° to 83° B. gravity in this way are situated 30 miles from the blending stations of the buying companies. These are unusual examples, but a number of plants ship their product 6 to 10 miles in tanks of this character. Generally the tank is kept under a pressure of 10 to 20 pounds in order to reduce losses of the light condensates from agitation and heating during transportation.

TRANSPORTING CONDENSATE IN CRUDE OIL.

Certain large producing and refining companies in California which buy or produce casing-head gasoline gage the product in the storage tanks at the compression plants for settlement, and have the condensate pumped directly into their crude oil storage or pipe lines leading to their refineries. They not only recover the gasoline when the crude oil is refined, but take advantage of the fact that the condensate "cuts" the crude, making it flow through the lines more easily. The extremely light fractions that are thus injected into the crude oil, and will not condense in the usual refinery condenser boxes, can be and are in many refineries compressed and cooled as in compression plants. This product is immediately blended at the refinery and thus held and sold.

As previously mentioned, a company in the Mid-Continent field produces only such condensates as can be shipped in tank cars. This is done by regulating the pressures and the temperatures used in the compression plant, so that only such condensate as can be shipped unblended will be precipitated.

A plant in California and some plants in other fields reduce the unblended condensate to conform with the shipping rules by weathering, because of the cost of shipping in blending naphtha. The condensate is exposed in storage tanks to atmospheric temperature and pressure until the vapor tension is reduced to the desired point, and then shipped in insulated cars. At times warming with steam is resorted to if the atmospheric conditions do not bring about the proper results. The use of steam is not to be recommended, however, except when absolutely necessary, because of the loss of some of the heavier fractions with the lighter ones.

METHODS OF BLENDING.

Blending condensate with the various distillates used for that purpose, as practiced at present, is done at times in stages, and at many different points in the precipitation, storage, or transportation of the product.

The product of plants that ship their condensate without being blended usually goes to refineries or blending stations belonging to purchasers of this type of product, who blend the condensate before sending it to the retail markets. One blending company in West Virginia buys condensate, pumps it to the plant in pipe lines, stores it in closed tanks until needed, then blends it with naphtha in the following manner:

A tank car of naphtha is one-half unloaded, usually into an empty tank car, and then condensate is slowly pumped in through a valve in the bottom until the tank is filled. The condensate rises through

the naphtha and slightly agitates it, and in this way becomes absorbed and blended with the naphtha. At times the operation is reversed, a tank car being half filled with condensate and the naphtha pumped in from above. No further treatment is used, the car being shipped as soon as filled. The agitation during shipment tends to complete blending if such is necessary.

Blending practice at some refineries and casing-head gasoline plants is practically the same as described above except that stationary tanks are used in place of tanks on cars. At other blending plants a pump is used in blending, its suction being connected with two tanks, one of blending stock and the other of condensate. The flow of each is regulated in the pipe line by valves, the discharge of the pump going to a storage tank. From time to time the mixture in the storage tank is tested for gravity, if the blend is too light or too heavy the flow of either naphtha or condensate is adjusted to give the desired mixture.

At some plants methods of blending are more complicated. The procedure used by one company receiving its condensate by auto truck is as follows: A given quantity of California distillate with a gravity of 53° to 55° B. is placed in a cone-bottom blending tank, where it is washed with acid solution, caustic solution, and water; after this treatment condensate with a gravity of 72° to 82° B. is forced into the tank from the bottom. Air is then blown through the mixture to agitate it and remove the lightest fractions of condensate and dissolved gases. The mixture is tested for specific gravity, after which enough still-run California gasoline with a gravity of 58° B. is added to bring the whole to a gravity of 60° B. The blended gasoline produced by the above method and ingredients is sweet, water white, and has the following characteristics: 5.9 per cent distills over up to 140° F., the distillate having a gravity of 79.9° B.; 70 per cent distills over up to 246° F.; and 30 per cent distills over between 246° and 344° F.

Distilling this blended product in 5 per cent cuts shows it to be an exceptionally good motor fuel with none of the usual fractions missing.

While holding condensate in storage some blending companies and refineries, as well as compression-plant operators, keep the tanks containing such stock under pressure and often the tanks are insulated or housed and shaded in order to reduce evaporation by the sun and the atmosphere, one company having gone to the expense of building a louver tower over the tanks and keeping small sprays of water constantly covering them. The tanks, which are held under pressures of 10 to 20 pounds, are set in a concrete or wooden basin. The water collects in this basin, thence it is again circulated over the tanks by pumps. No outside cooling of the water is resorted to, the

evaporation from falling through the tower and over the tanks being sufficient to keep the water at a temperature considerably below that of the atmosphere, also any volatilization of the condensate itself tends to cool the liquid.

Where blending is done by refining or blending companies, as has been described, the operation is complete and the blended product is ready for market. The blended gasoline made in the Eastern fields has a gravity of between 65° and 70° B., and that made in California between 58° and 63° B. The difference in gravity is due to differences in the character of the crude oils from which the condensate and the blending stocks were made; this variation is discussed in later paragraphs. The blending is usually done so as to bring the product to a given gravity by the mixture of the two ingredients, whereas the final end point is determined by, and is the same, as the end point of the naphtha used. Although the proportions vary under these conditions, the proportions to be mixed to make a product of a given gravity can be approximately calculated from the gravity and the end points of the two liquids.

The methods of disposal of condensate mentioned are found mostly in California and some eastern districts; with few exceptions the practice in the Mid-Continent field is to blend either at the compression plant or at the loading station operated in conjunction with the plant.

BLENDING BY PLANT OPERATORS.

When the blending is done by the operator of the compression plant, one of the following methods is generally adopted: (1) Blending all of the condensate at the blending station or loading racks; (2) blending to a given stage at the plant, and transferring the partly blended product to the loading racks and either finishing the blending there or shipping it to some point at which naphtha is cheaper or more readily obtained; and (3) completing the entire operation at the plant.

BLENDING AT THE LOADING RACKS.

Many of the companies controlling two or more compression plants, situated in the same field and tributary to the same shipping point on a railroad, have adopted the method of blending the products from all their plants at a central station. A centrally situated loading station with racks and tanks (see Plate XII, A, p. 52), is necessary in any event for storing and loading the plant products and unloading and storing the blending stocks, so that the stations can be also fitted for blending practically without additional cost of labor and small increases in tankage.

In stations used in this way, the usual equipment consists of tanks of any desired capacity for the storage of naphtha, other tanks for

the storage of condensate, and tanks for the blended stock and from which the marketable product is transferred to tank cars for shipment.

The plant condensate is pumped or forced by its own pressure through small pipe lines into the condensate storage tanks, and the naphtha from tank cars is pumped into the naphtha tanks. From the naphtha storage tank a given quantity of naphtha is first pumped into the blending tank, then condensate is forced into the blending tank at the bottom, being allowed to rise through and be thoroughly absorbed by the heavier naphtha. When the predetermined quantity of each stock has been mixed in the blending tank, samples are tested for gravity, and if any change in gravity is desired more of the heavy or light stock is added, as the tests indicate to be necessary. The blended gasoline is next tested for vapor tension, and, if it found to be too high for shipment in the cars used (standard or insulated tanks), the tank is allowed to stand open to the atmosphere for several hours, or even days, if necessary. In some plants it has been found necessary to heat the blended products to 80° to 90° F. with steam, in order to reduce the vapor tension to the required point in a limited time, so that the blending could continue without the installation of an excessive amount of tankage. Heating with steam should not be resorted to if avoidable, as a quick rise in temperature drives the lighter condensates off rapidly, carrying with them portions of the heavier and less volatile fractions. When the specific gravity and the vapor tension have been brought to a point within the regulations governing shipments of gasoline, the contents of the blending tank are either drawn off into cars for immediate shipment or pumped into a storage tank.

The blending method used at the majority of such stations usually consists merely of mixing calculated quantities of the different stocks to be disposed of and reducing the vapor tension of the mixture by open contact with the air or by slight warming with steam coils, placed in the blending tank.

The advantages of blending at loading and storage stations are that the productions of a number of plants can be handled, close connection with railroad service, and the low cost of handling both the naphtha and the blended stock. At times the tank cars are used as blending tanks, as described in a previous paragraph.

PARTIAL BLENDING AT PLANT.

Some operators, because of the plant being in a place where it is difficult or costs too much to bring in large quantities of naphtha, or, more often, because the company controls a refinery and desires to refine the gasoline by further treatment, such as distilling, have adopted the practice of blending only so far as is necessary for shipment at the compression plant or blending station, the final blending

and treatment being given at the refineries or points where the desired quantities and qualities of distillates may more readily be obtained. Some operators blend partly at the plants and finish the operation at the blending station or loading racks as described above, thus saving the costs of handling, of pipe line, and of the pumping capacity necessary to transfer all of the heavy blending stock to the plant and back to the loading station.

When the blending is completed at the blending station, it is done in the way described, except that smaller proportions of heavy naphtha are added, because some heavy naphtha has been previously added to the condensate at the plant.

PLANT BLENDING METHODS.

Plants at which blending is entirely or partly completed have developed methods and practices quite different from the usual way of mixing the two constituents in a blending tank. Many plants still blend the condensate and the naphtha in storage or blending tanks, using the methods previously described above, but a tendency has developed for blending at much earlier stages of the process.

BLENDING IN "MAKE TANKS."

The first step in this development was when certain operators pumped blending naphtha into the tanks known as "make tanks" (see Pl. IV, *C*, p. 26, and XII, *C*, p. 52), which receive the condensate from the accumulator tanks, and in which the total make of one day or shift was measured before being transferred to storage. The method as now used is to pump naphtha into the tank at such a rate that the percentage in the mixture would be somewhat below that desired in the final blend, or to place a given quantity of naphtha in the make tank and discharge condensate from the accumulators into the naphtha. In transferring condensate from the accumulator tank to the make tank, the sudden release of pressure causes violent weathering or boiling, owing to the high vapor tension. By adding heavier blending stocks at this point the vapor tension is lowered, with consequent lessening of losses from the light condensate and dissolved gas weathering rapidly and carrying off with them part of the heavier fractions.

The product of this blend is gaged in the make tank, the quantity of naphtha deducted, and the actual plant production calculated. At the end of each day or shift the mixture is transferred to storage tanks and the blend completed or shipped to another point for blending as described above.

BLENDING IN ACCUMULATOR TANKS.

Blending in accumulator tanks, as found in practice by the writer, consists of pumping naphtha slowly and continually into the tanks, connected with the high-pressure coils, at a pressure a few pounds in

excess of that at which the gas is being treated. The naphtha is injected into the tank through small ($\frac{1}{4}$ -inch) pipes at a point near the top and through fittings which cause a spray, the theory being that the spray will collect fine particles of condensate in falling through the gas and reduce the gravity and the vapor tension of the product before it is released from the high pressure at which it is precipitated and thus reduce losses of condensate.

Operators using this method claim it produces a marked increase in plant production, one in particular claiming a net increase of 10 per cent. The mixture is drained from the accumulator tank as in other practice, either automatically or by hand, as the custom of the plant may be, the quantities being figured as previously described to determine the plant production.

HOT BLENDING.

When naphtha is injected into the high-pressure gas while it is in the coils, or before it has reached the coils, and is still hot from compression, the method is called "hot blending."

This method has been adopted at some plants where the gas treated contains large proportions of the exceedingly light fractions and the condensate had shown extreme losses in storage and during weathering and blending by other methods. A Pennsylvania operator producing condensate with a gravity of 92° to 95° B. claims a net gain of 15 per cent in marketed condensate from the use of this method, and one in Oklahoma, approximately 30 per cent.

Hot blending has been tried out at two California plants, in different fields, to the writer's knowledge and no advantage gained. The plants produce condensate with a gravity of 82° to 86° B., using pressures between 200 and 250 pounds.

One eastern plant, which compressed the gas to 300 pounds pressure, injects naphtha through a needle valve placed in the high-pressure water-cooled coil header at the intake (hot) end of the coil. The naphtha is pumped through the valve at a pressure of 400 pounds per square inch, which causes it to spray or atomize in the header and intimately mix with the hot gas as the gas divides into the coil pipes leading out of the header. The first few (10 to 15) feet of this coil is kept dry to permit the hot gas to vaporize as much of the naphtha as possible before the gas and the naphtha are cooled by the water sprayed over the rest of the coil.

What happens to the injected naphtha is not definitely known, but it appears that the naphtha is divided into three parts. One part is volatilized by the heat of the high-pressure gas (190° F. on the day of the writer's visit), carried with the gas into the water-cooled coils, and again condensed, thence it is carried to the accumulator tank with other condensate. A second part of the atomized naphtha is probably carried mechanically into the upper pipes of

the coil by the flow of gas, where it settles out, owing to the slower rate of flow, and carries with it other condensable vapors. The third part, which is neither vaporized nor carried mechanically into the upper members of the coil, goes to the bottom of the header and flows with the gas through the bottom pipe and blends with the condensate and naphtha in the discharge header, while still under maximum pressure. The mixture then flows to the accumulator tank and thence is trapped into storage tanks.

The quantity of naphtha injected into the header is calculated to be somewhat less than is needed to bring the mixture to the gravity desired, the balance being added, in this plant, to the partly blended stock in the make tank, which is also held under pressure. Another feature of the practice at this plant is that the pressure on the blended stock is reduced slowly to avoid violent boiling or weathering. This is accomplished by holding a pressure of 50 or 60 pounds on the make tank while the output of the plant is being transferred to it during a day or shift. At the end of that period another tank is put into service and the pressure on the tank containing the day's "make" is slowly relieved, while the stock gradually acquires approximately the temperature of the atmosphere. At this point the blend is transferred to storage tanks or placed in tank cars for shipment. For each 100 gallons of condensate produced 77 gallons of naphtha is pumped into the coils, which lowers the gravity of the condensate from 90° to 96° B. to between 70° and 76° B.; later, in the "make" tank, enough eastern naphtha, with a gravity of 58° to 60° B., or California distillate, with a gravity of 48° B., is added to form a blend of the desired quality.

At another plant a practice similar to the one just described was adopted, except that the naphtha was injected into the hot-gas line at a point just beyond the oil separator and 40 or 50 feet ahead of the coils, the naphtha thus traveling with the gas through the coils and on to the accumulator tanks. This plant, although making 9 gallons of condensate per thousand feet of gas treated, as measured in the accumulator tanks, had by the old method of blending been able to market only 2.5 gallons. By the adoption of hot blending it was able to market 3.7 gallons from each thousand feet of gas treated. The loss, however, is still extremely high, and it is probable that an entire readjustment of pressures and temperatures throughout the plant would show better returns and smaller losses.

POSSIBLE IMPROVEMENTS IN BLENDING METHODS.

No set of rules can be given or standard methods of blending described that would even approximately cover all conditions, but there are methods and practices that if adopted by individual plants would have marked advantages and show a decided saving of condensate.

The problems of each plant must necessarily be taken up separately and a study made of the properties and characteristics of the condensate formed, such as the gravity, the vapor tension, and the proportions of the different "cuts" and the hydrocarbons they contain. That these vary widely is shown by the different pressures and temperatures necessary to precipitate the condensates, also by the vapor tensions or wildness of different plant products. The point at which blending is done should be given attention, to find whether the best results are obtained by blending in storage tanks, in make tanks, in accumulator tanks, or in coils, also the effects of release of pressure and rise in temperature should be studied to find whether sudden lowering of pressure and rise in temperature does not cause undue losses. Blended products are at many plants placed in storage tanks which are not protected from the sun and a quick rise in temperature is to be expected. The color an exposed tank is painted, has a direct bearing on the amount of heat absorbed from the sun's rays and taken up by the contents. Releasing the pressure on the condensate slowly, or holding it under a low pressure and storing it in insulated tanks has, at a number of plants, been found to decrease losses.

Refining companies have found that storing the lighter distillates in tanks with the water-sealed top, painted glossy white, show reduced losses well worth the expense of construction and operation of this system of storage.

LOSSES OF CONDENSATE IN WEATHERING AND SHIPPING.

Plant operators, in reporting losses of condensate, often use as a basis the total amount of condensate collected and measured in the accumulator tanks before blending, and while still under pressure, which is obviously the wrong place to make such an estimate, because such measurement is taken during treatment and not at the final stage of production. The condensate in the accumulator tank, because of the temperature and pressure, contains some dissolved gas and fractions of the hydrocarbon group which it is impossible to hold under normal atmospheric conditions, and which should not be counted as loss when computing the plant production.

The writer, in order to form an accurate idea of the losses at different plants, and as far as possible to use the same basis in estimating these losses, has obtained from the plants listed the quantity of condensate actually stored in tanks, either blended or unblended as the practice indicated. With this quantity as a basis estimates of the losses in further operations, such as weathering, pumping to loading stations, loading in cars and shipping, are computed.

In Table 2 (p. 29), the column headed "Daily production, gallons," represents the number of gallons that were actually marketed by the different plants listed, as nearly as these figures could be arrived at, and is net production after all losses are deducted.

CALIFORNIA PRACTICE.

California plants, in general, trap or blow the product of the accumulator tanks directly into storage tanks without blending and measure the make in the storage tanks. As these tanks are seldom held under pressures greater than 5 to 15 pounds, weathering goes on continually and causes an unknown loss, only the weathered product being measured.

At plants where the raw condensate is pumped from the storage tanks into the crude oil storage tanks of purchasing companies, under this method of calculating losses the results show no loss except that due to weathering while the condensate is being held in plant storage, the product being sold on the gage readings taken in the storage tank at the compression plant.

Companies shipping raw condensate in automobile trucks distances of 2 to 40 miles report losses in loading, transportation, and unloading of 2 to 7 per cent. Those delivering their product through pipe lines 1 to 40 miles long report losses of 1 to 6 per cent, depending on the length of the lines, the amount of leakage from them, and the pressure necessary to force the liquid through.

EASTERN PRACTICE.

Many plants in the eastern fields ship raw condensate by pipe line and trucks, as in California practice, with losses the same or slightly greater in amount than those quoted, owing to the raw products having higher gravities and vapor tensions. One large oil company has recently laid pipe lines to a number of the fields in northern and western Pennsylvania for the purpose of collecting the raw condensate produced in those fields and which was formerly hauled to market in trucks. Well-constructed pipe lines will undoubtedly minimize losses from handling condensate produced in this district.

OKLAHOMA PRACTICE.

Casing-head gasoline producers in Oklahoma estimate the total loss as varying between 10 and 40 per cent. The losses during weathering of the blended product run from 10 to 20 per cent, depending on the gravity of the raw product and the vapor tension of the blend. Often heating with steam is necessary, before loading the condensate into tank cars, to bring the vapor tension within the limits required by the railroad shipping rules, especially in cold weather, when weathering will not bring the blended stock to the desired condition.

The loss in this field due to transferring the product from the plant storage tanks to loading or blending stations varies from 2 to 7 per cent, and in loading from a rack to cars, between 1 and 5 per cent.

The loss during shipment in standard tank cars is variously given as being between 2 and 10 per cent, with an average of approximately 6 per cent, the loss depending much on the distance and length of time used in haulage, the atmospheric temperatures encountered, and the condition of the tank used for shipment.

The so-called thermos cars, an insulated tank car, have given very satisfactory returns on their cost to certain producers using them, who report a maximum outage (loss in transit) of 2½ per cent when used for long shipments in hot weather, and report instances of no outage on short hauls in moderate, rainy, or cold weather. The railroad rules governing shipments in insulated cars are more in favor of the shipper in regard to vapor pressures, allowing 15 pounds as the maximum in place of 10 pounds as in standard cars.

The losses in blending and shipping in the Mid-Continent field may be generalized as follows:

	Per cent.
Weathering.....	5 to 20
Transferring to loading station.....	2 to 7
Loading in car tanks.....	1 to 5
Shipping in standard cars.....	2 to 10
Total losses.....	10 to 42

Plants treating gas that contains large proportions of the lighter fractions and using pressures and temperatures that will precipitate these fractions produce the "wildest" condensate and in consequence suffer the greatest losses. As the converse of this practice a single-stage plant (No. 76 in the tables) in the Mid-Continent field, mentioned in several places throughout this paper, produces a condensate having a gravity of 79° B. and a vapor tension of about 5 pounds that is shipped in standard tank cars with a total loss of 2 to 3 per cent, including the losses from being transferred several miles through pipe lines to the loading racks and from loading into the cars.

NAPHTHA USED AS BLENDING STOCK.

To form an ideal motor fuel, the distillate or naphtha used in blending should be one that will give the mixture a uniform series of fractions between the temperatures at which distillation begins and finishes, with none of the fractions with boiling points so high as to cause incomplete combustion and carbon deposits in motor cylinders.

To blend and weather condensate to correspond with the above conditions is possible, but such practice, because of the great waste and expense, is followed only in blending special gasolines for engine or speed tests. Also, in making straight refinery distilled gasoline, because of the increased demand for motor fuel, more and more of the heavier distillates have been cut into the motor-fuel fractions, causing a lower gravity and a higher end point.

Rittman, Dean, and Jacobs ^a have shown clearly the differences in still-run and blended gasolines (see fig. 13) as they are put upon the market. They state that the blended casing-head products have larger percentages distilling below 50° C. but have longer distillation ranges, which tend to make the slope of the temperature-percentage curves for these gasolines flatter than those of straight refinery products. They also state that any gasoline having an unusually large distillation cut below 50° C. and with considerable percentages distilling within the temperature ranges of 150° to 175° C. and 175° to 200° C., and being deficient in constituents boiling at intermediate

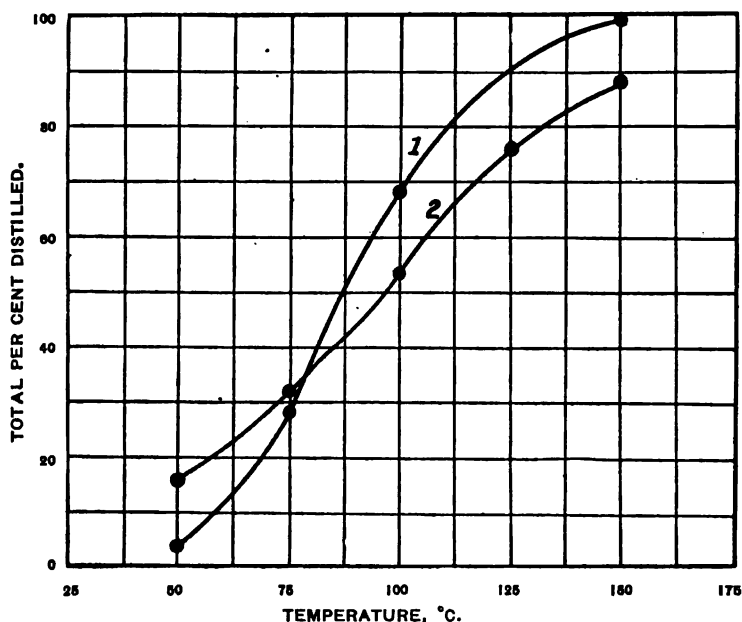


FIGURE 13.—Curves showing volatility ranges of refinery and of casing-head gasoline. 1, refinery gasoline; 2, casing-head gasoline. The flatter slope of the curve for casing-head gasoline shows that the content of both low and high boiling constituents is greater in the blended gasoline. After Rittman.

points of the distillation, may be classed as one of these blended products.

The naphthas or distillates being used for blending are those fractions that distill from crude oil after the cuts marketed as "straight still-run" gasoline have distilled off. The naphthas made from eastern and Mid-Continent crudes range in gravity from 46° to 60° B., whereas those made from the asphaltic base California crude oils range between 42° and 52° B. The difference in the eastern and western distillates is due to the fact that the crude oils in the different fields differ in character, having paraffin, asphaltic, or mixed bases.

^a Rittman, W. F., Dean, E. W., and Jacobs, W. A., Physical and chemical properties of gasolines sold throughout the United States during the calendar year 1915: Tech. Paper 163, Bureau of Mines, 1916, p. 27.

The eastern and the California naphthas used in blending have approximately the same end points, although the gravities differ 7° to 10° B.; also the blends formed have different gravities, although the end points, power developed per gallon, and the completeness of combustion of the mixtures are practically similar. The differences in the specific gravities of the various cuts of similar end points from the different crudes decreases as the cuts become lighter.

Mid-Continent crudes having mixed bases vary between California and Pennsylvania crude, and have gravity and end point ratios between the two extremes stated above.

Certain companies in buying blended gasoline specify that the product shall have a gravity of not less than 67° B. and an upper end point not more than 400° F., which will require that naphtha with a gravity of about 55° B., if from Mid-Continent crude, or 48° B., if from California, be used in making the mixture.

Numerous blending plants, however, use distillates with as low gravity and as high end point as the kerosene fractions, and, as far as is known, find no trouble in marketing such blends. In using naphthas for blending it has been found that the heavier naphthas give better results than the lighter ones in lowering the vapor tension of the mixture, for equal quantities put in, the heavy fractions having a tendency to "hold down" the light fractions.

PROPORTIONS IN BLENDS.

In general it may be stated that blended gasoline usually consists of a mixture of half casing-head gasoline and half naphtha or distillate, but the proportions vary, depending upon the gravity and vapor tension of both constituents, blending being carried to a point, in conjunction with weathering, that brings the product within the shipping rules and shows maximum profits to the producers.

STRATIFICATION OF BLENDED GASOLINE.

In many quarters belief has been expressed that in blended gasoline the light condensate fractions separate from the heavier distillate fractions, causing stratification. The writer could find no direct evidence that such stratification takes place, and interviewed many operators and blenders who had made various tests and were unable to find such a condition. It is true that owing to change of temperature, in a closed tank, the lighter fractions at times vaporize and condense on the sides of the container and drain down, floating in a thin layer on top of the liquid. This condition may have given rise to the belief regarding stratification, but, as shown, is not due to separation of the two or more blended constituents through differences in their gravities. All fractions of petroleum oils are gen-

erally considered as soluble, one in the other, and a blend of two or more fractions such as naphtha and condensate should not separate or stratify. All evidence obtainable indicates that no such stratification actually takes place in blended motor fuels through the difference in specific gravity of the members blended. A test made on a California blended product and reported to the writer showed no separation. The blend, which was a small quantity of condensate with a gravity of 105° B. and a large quantity of distillate with a gravity of 48° B., was placed in a 10-barrel tank and tested after standing one week without being disturbed. Samples drawn from the top and bottom had specific gravities differing only 0.07° B. This blend found a ready market as fuel for motor trucks.

The naphthas used and the blends marketed at the present time depend to no small extent on the market conditions of the heavier distillates. Naphthas that can be obtained in quantity easily and regularly, thus insuring a steady supply, are often chosen in preference to a more perfect blending stock, supplies of which can not be depended upon.

COSTS OF COMPRESSION PLANTS.

The widely varying conditions under which compression plants are being installed, both as to situation and the machinery and the steel markets, make estimates of the costs of plants so uncertain that the subject will be undertaken with the idea of showing the costs of plants recently installed, of which the writer has knowledge, rather than to attempt to estimate costs in general for present or future installations.

An operator in the Mid-Continent field who has built and is running three plants computes plant construction costs on a unit basis and gives the costs of one unit that will treat 400,000 to 500,000 cubic feet of gas daily, as follows: One single-stage unit, compressing the gas to a pressure of 75 or 90 pounds, \$8,000 to \$9,000; one two-stage unit compressing to 200 or 250 pounds, \$15,000 to \$18,000. Vacuum pumps at the compression plants are included in this estimate, but not gathering lines with their pumps, or the expansion engines that are used at the plants. On this basis, a 1,000,000-foot plant compressing gas to a pressure of 250 pounds would cost approximately \$36,000, or \$36 per 1,000 feet of capacity.

The expense of construction and equipment of a plant compressing 1,250,000 cubic feet of gas daily to 250 pounds, put into commission in June, 1916, was stated to be \$36,570, divided as follows:

Cost of compression plant with a capacity of 1,250,000 cubic feet.

Machines, engines, compressors, water pumps, and air pumps and tanks.....	\$18,240
Pipe and fittings, engine and compressor connections, cooling coils and double-pipe coils.....	1,534
Building compressor room with steel frame, cooling tower, accumulator tank and storage housing, auxiliary engines and pumps.....	6,756
Gathering lines.....	3,797
Tankage.....	1,105
Gas traps.....	700
Electric plant.....	395
Labor, carpenter work; pipe-fitting; concrete, etc.; setting engines.....	4,043
Total.....	36,570

No expansion set is included in this estimate, but when the capacity of the plant was doubled later in the same year a simple single-stage compressor was installed as an expansion engine and is used to expand all of the gas treated, or 2,500,000 cubic feet. The construction of the first half of the plant shows a cost of \$29 per 1,000 feet capacity. The entire cost of the plant after the original capacity was doubled and an expansion engine installed is approximately \$60,000, or \$24 per 1,000 feet capacity.

Another plant with a total capacity of 2,500,000 feet, built during the same period, using the same pressures and very similar to the plant described above, in units and "hook-up" (compressor and engine drive), but using a drilling engine as an expander, cost \$55,000, or \$22 per thousand feet capacity.

The cost per 1,000 feet of capacity of plants treating small quantities of gas, 100,000 to 250,000 cubic feet daily, is relatively higher; instances having come to the attention of the writer of costs of \$40 or \$50 per 1,000 feet capacity for plants of that size.

One small plant—a 250,000-foot plant compressing gas to 250 pounds—including an expansion engine, cost \$11,000, or more than \$44 per 1,000 feet of capacity.

Plants treating small quantities of gas under pressures obtained by single-stage compressors and having simple cooling systems, such as are found in many eastern fields, notably Sistersville, W. Va., cost much less than is indicated by the above estimates.

No estimate of costs of gathering systems, with vacuum stations and booster compressors, can be made, as the conditions vary from a few hundred feet of 6-inch lines to lines of 8 and 10 inches diameter extending 5 to 10 miles from the plant and having as many as 30 pumps and "booster" machines forcing the gas through the mains and maintaining low pressures on the wells.

MINIMUM PLANT CAPACITY AT WHICH GAS CAN BE TREATED PROFITABLY.

The question of how much gas is necessary to make the installation of a compression plant profitable often arises. This subject is approached more easily and clearly from the point of production, the quantity of gas being allowed to vary accordingly. Operators owning wells producing enough gas to produce 150 or more gallons of condensate daily can at the present price of that product well afford to install a plant, provided that no royalties have to be paid, that the gas will yield 1 gallon or more per 1,000 cubic feet, and that not more than one extra man must be kept on the pay roll. To produce this quantity of condensate under these conditions, 150,000 cubic feet of 1-gallon gas, or 50,000 cubic feet of 3-gallon gas would be profitable. In plants of this size owned and operated by oil and gas producers, often no extra help is needed, the plant being watched and taken care of by pumpers in making their regular rounds of the lease. The cost of such plants varies under average conditions between \$4,000 and \$8,000, and profitable returns on the costs of installation are shown.

With these costs as a basis and considering royalties and additional labor to run the plants, the quantities and richness of gas necessary to make an installation profitable may be estimated. Many small plants in the eastern fields are treating 75,000 to 100,000 cubic feet of gas daily, and the fact that they continued to run during times when condensate sold at 8 and 10 cents per gallon would indicate that profits under the present conditions are gratifying. The price received in August, 1916, at Sistersville, W. Va., for condensate with a gravity of 86° B., was reported to be more than 18 cents per gallon.

PLANT RECORDS.

The writer in collecting detailed plant data found that many, in fact most compression plant operators used no system of daily report sheets or records of plant operation further than making each day an estimate of the condensate produced. It is difficult to see how plants of this character can be operated intelligently, or improvements made, unless accurate records of all the more important plant functions are carefully kept and used in directing changes and experiments to improve the practice and increase the quantity and quality of the product. At fully one-half of the plants visited no attempt was made at keeping data of any kind excepting records of the production, and only a limited number of those keeping accurate and detailed records made any use of them; or had tried to improve either the operation or the product. For instance, the pressure of 250 pounds is almost universally used as the ultimate pressure at plants throughout the western fields, because other operators use

it and because the machines and pipe fittings are designed for that pressure, the actual qualities of the gas being treated not being considered. At one plant that had been using a pressure of 250 pounds, it was found that the engines gave less trouble when the pressure was reduced to about 200 pounds and that no less condensate was produced, showing that the higher pressure had not been used because it was best suited to the gas but because it was standard for the machines in use.

In every plant the best pressures and temperatures to use must be found by trial and experiment, but without records of results and conditions improvements will be slow in coming. The accompanying form shows the daily report sheet used by a progressive California operator. Many of the records are made by recording gages and thermometers, the charts being filed for reference and comparison with the report sheet. At any sign of a disturbance of any plant function, additional information is gathered, such as the specific gravity of the gas before entering and after being discharged from the plant and after each step in its treatment. The quantities of condensate precipitated and collected at each accumulator tank and its specific gravity are also found useful in locating plant disturbances or irregularities. Records of temperatures of the gas, at each change of pressure and after each set of coils is passed or each fraction of condensate is removed, should be kept. Records of cooling water and atmospheric temperatures, in conjunction with other records, may show either the cause or the effect of changes in the gas or in plant conditions. Although the cooling of gas in coils and of water in towers has been practiced for years, many casing-head gasoline plants have installed only the most primitive and least efficient systems, and the lack of records and data obscure the fact that poor results are being obtained from their use.

Records of engine speeds and other purely mechanical data, such as the number of hours machines are shut down and the reasons for such delays, are of use in determining costs, plant efficiency, and the efficiency of men in charge of plants or the machines, as well as the principal causes of trouble and weakness of any machine or part.

Accurate plant records are also valuable in cooperating with operators of other plants in an endeavor to improve general plant practice and operation. It was noticed in districts where friendly cooperation existed that the practice in general in that district was more modern and that the operators and men in the plants had a broader understanding of the process as a whole.

The measure of success in most compression plants has been considered only from a narrow viewpoint, the improvements possible and the waste of condensable vapors contained in the gas being overlooked.

Report sheet used at compression plant in California.

DAILY REPORT. On tour..... Date.....
 Gasoline Plant C, No. 1. First engineer..... Third engineer.....
 Off tour..... Second engineer..... Fireman.....
 Gas from wells No.....

MACHINE REPORT.

SHUT DOWNS.		Number of machine.										OIL USED—GALLONS.														
		1		2		3		4		5							6		7		8		9		10	
Hours.	Min. utes.	Hours.	Min. utes.	Hours.	Min. utes.	Hours.	Min. utes.	Hours.	Min. utes.	Hours.	Min. utes.	Hours.	Min. utes.	Hours.	Min. utes.	Hours.	Min. utes.	Hours.	Min. utes.	Hours.	Min. utes.	Compre- sion and gas cylinder.	Glycerin.			
For shift.....																										
For month forwarded.....																										
Total for month.....																										
Machine No.....		1	2	3	4	5	6	7	8	9	10	For shift..... For month forwarded..... Total for month..... Total for year.....											Steam cylinder.	Journal.	Compre- sion and gas cylinder.	Glycerin.
Time of reading.....		3										For shift..... For month forwarded..... Total for month..... Total for year.....											Steam cylinder.	Journal.	Compre- sion and gas cylinder.	Glycerin.
		6																								
		9																								
		12																								

GAS TEMPERATURES—DISCHARGE FROM LOW AND HIGH PRESSURE CYLINDERS.

Machine No.....	1		2		3		4		5		6		7		8		9		10	
	Inter- medi- ate.	High.	Inter- medi- ate.	High.	Inter- medi- ate.	High.	Inter- medi- ate.	High.	Inter- medi- ate.	High.	Inter- medi- ate.	High.	Inter- medi- ate.	High.	Inter- medi- ate.	High.	Inter- medi- ate.	High.	Inter- medi- ate.	High.
Time of reading.....	3																			
	6																			
	9																			
	12																			

TEMPERATURES AND PRESSURES.

	At- mos- pheric tem- per- ature.	Cool- ing water.	Scrubber.		High-pressure Inlet.		Intermediate expansion.		Discharge to lease.		Best. coil No. 1.		Coil No. 2.		Coil No. 3.		Number of coils cooled by low-pressure expansion.	Number of coils cooled by intermediate expansion.	Number of coils cooled by low-pressure expansion.
			Pre- sure.	Tem- per- ature.	Pre- sure.	Tem- per- ature.	Pre- sure.	Tem- per- ature.	Pre- sure.	Tem- per- ature.	Inlet.	Out- let.	Inlet.	Out- let.	Inlet.	Out- let.			
Time of reading.....	3																		
	6																		
	9																		
	12																		

EXPANSON SET—TEMPERATURES AND PRESSURES.

PRODUCTION REPORT.

	Gal- lons.	Grav- ity.	Gal- lons.	Grav- ity.		
Low pressure.....					For shift..... For month forwarded..... For month total..... For year forwarded..... For year total.....	
High pressure.....						
Coil 1.....						
Coils 2 and 3.....						
Exhaust.....						
Boilers cleaned..... Boilers repaired..... Furnaces repaired..... Water used.....						

REMARKS:

Certified correct by.....First Engineer.
Sheet No.....

The staff of the Bureau of Mines receives many letters asking information regarding the manufacture of gasoline from natural gas, which they willingly and gladly answer, but in most of these letters the data regarding both the gas and the operation of the plant are so meager that answers must be of a very general character and are probably as unsatisfactory to the correspondent as to the writer.

SUMMARY.

In the following paragraphs some of the more important factors relating to natural-gas gasoline and its production by the compression process are briefly discussed.

TESTING NATURAL GAS TO DETERMINE ITS SUITABILITY FOR MAKING GASOLINE.

The various hydrocarbons, and also the impurities, found in natural gas as it comes from oil or gas wells, have a direct bearing on the plant practice and treatment that will yield the most condensate. The more complete and thorough the tests made on the gas to determine its composition and physical characteristics are, the less chance there will be of those interested building and operating a compression plant that is not best suited for treatment of that particular gas.

Thus far the plants installed have been generally of standard design and equipment using the maximum safe pressures for which standard machines and fittings were built and temperatures obtainable by simple methods of water cooling, little consideration being given to the qualities of the gas to be treated.

Gas testing as followed at the present time is done only to determine if the gas can be profitably treated, and, if found so, a plant of general standard design is put in operation. It must be admitted that these plants as a whole, throughout the United States, have proved financially successful, but as a rule such success is due, in the writer's opinion, more to the fact that only the rich gas has been taken for treatment than to careful plant design and operation. The great increase in production at a few of the plants where the operators have studied the gas being treated and installed equipment best suited to its composition and characteristics indicates that gross waste is taking place at many plants of standard design. As more is learned of the best methods of treating natural gas in compression plants, gas of lower gasoline content can be and will be used in such plants.

At many plants visited no gas tests of any kind had been made since the original tests to determine whether the gas was rich enough in gasoline to warrant treatment. With the aging of the wells, the extension of gathering lines, and the installation of vacuum pumps, which often draw air into the gas lines, it is hard to believe that no variation in treatment was necessary if the best results were desired.

Gravity tests of the gas used should be made from time to time at all plants. The results will indicate any changes in composition and usually whether air has been taken into the line. This test, if made at different points in the treatment and on the treated gas may also indicate either a change in the character of the gas or that some part of the plant is not working as usual. Other tests, such as absorption and compression tests and analyses, made and recorded at regular intervals, have been of value in determining sources of trouble and indicating the need of experiments and of plant changes.

Experimenting with the entire plant or one complete unit by changing the temperatures and pressures and recording the results may lead to better recovery or a better product.

PRESSURES AND TEMPERATURES.

Gas from wells that are being and often have been gas-pumped for years, and are held under high vacuums is composed largely of the heavier vapors that would, unless vacuums were used, remain as liquid in the oil, and in treating such gas extreme pressures and temperatures are not necessary.

In eastern fields, where these conditions are most often found, single-stage plants working at a pressure of about 100 pounds and using such temperatures as can be obtained in submerged coils cooled by the natural temperatures of well or creek waters, give satisfactory recoveries and produce condensates with a gravity and vapor pressure as high as can be handled, either blended or unblended. In the same fields, however, plants treating gas from old wells not held under vacuum have found that not enough of the vapor carried by the gas can be condensed at pressures lower than 300 pounds to be profitable. The condensate produced at that pressure was exceedingly wild and in order to effect a maximum saving required blending as early in the process of precipitation as possible. It appears from the above facts that gas taken from wells held under high vacuum carries portions of the higher-boiling fractions, distilled from the oil under reduced pressures, that would otherwise have remained in the oil in the sands. The removal of the lighter fractions by this method has less effect on the gravity of the refined oil than would at first seem probable, because a marked percentage of these vapors undoubtedly came from oil left in the sands which in all probability will never be extracted, and also because the oil production from many small wells held under high vacuums is stored for days, and even weeks, in tanks, exposed to changes in atmospheric temperature, during which time the lighter fractions are lost to a greater or less extent. Under these conditions, relieving the oil of its lightest vapors before it is exposed to evaporation, or while still in the sands underground, would save these valuable products.

In the newer fields, in which the gas is still produced under widely varying rock pressures, a maximum plant pressure of 250 pounds is almost universally used, and refrigeration has often been found to increase production 10 to 50 per cent.

Besides the usual cooling with water, at some plants the gas is also cooled in heat interchangers, while still at the maximum pressures used, with expanded gas. The dry compressed gas is expanded adiabatically in the power cylinder of a steam engine, its temperature being lowered at times to -100° F. In the heat interchanger the high-pressure gas is reduced to temperatures as low as -10° F., which causes vapors not condensed in the water-cooled system to precipitate. It is the writer's opinion that many plant operators are overlooking gas expansion as a means of increasing the net production of their plants.

The treatment of natural gas for gasoline is unlike those manufacturing processes in which the treated material may be stored and treated a second or third time after the first extraction or concentration of the desired portions, because the gas, after once coming to the surface, must be kept moving until treated and used as fuel or wasted to the atmosphere. Thus marketable fractions of condensate left in the gas after treatment are practically entirely wasted.

CONDENSATE.

The gasoline carried in natural gas and precipitated at different points in the treatment consists mostly of pentane and hexane, the fifth and the sixth members of the paraffin series, smaller proportions of heptane, the seventh member, and decreasing percentages, if any, of propane and butane, the third and fourth fractions. In condensates produced at high pressures and low temperatures, probably some propane and butane are present and with dissolved gas cause the high vapor tensions of some plant products. The amount of dissolved gas is probably of no importance, as far as volume is concerned, but there seems to be little reason to doubt that when the pressure on the condensate is relieved the gas has a decided tendency to cause boiling and agitation of the liquid, which, with boiling of the propane or the butane, causes losses not only of these constituents but also of some of the heavier members during weathering.

The gravity of the plant products as they come to the accumulator or the "make tanks" varies between 70° and 96° B., and the vapor tension, from 5 to 40 pounds.

The gravity and the vapor tension of the condensates as collected in the accumulator tanks of the successive stages become higher as the higher pressure and the lower temperature changes are reached. The products range from line drip or distillate with a gravity of 55° B., produced at atmospheric temperature and pressure, to condensate

with a gravity of 105° B., produced in the accumulator tank at the expansion-engine exhaust, at -40° F., the gas having been reduced to a pressure of 10 pounds.

Experiments and the equipment of recent plants indicate that to remove the condensate from contact with the gas as soon as possible after precipitation and collection is the best practice. This is accomplished by the use of small automatic traps, which drain the liquid from the accumulator tanks as soon as collected. The liquid then passes through pipes to "make tanks," or storage tanks, the pressure being reduced on a small quantity of condensate at each dumping of the trap with the least possible agitation and consequent boiling. This method reduces transfer losses and those from sudden lowering of pressure on large amounts of condensate at one time. The automatic traps are used in transferring either raw or blended products from accumulator tanks to storage or "make" tanks at lower pressures.

BLENDING.

Although some casing-head gasoline is shipped and used without being blended, most of it is mixed or blended with naphtha of lower gravity and vapor tension before reaching the consumer.

Condensate, although at times shipped unblended, is in the most modern plants and the latest practice blended as soon as possible after being formed, or even while in the process of precipitation. Some operators still ship their product partly blended or reduced in gravity and vapor tension to blending stations or refineries, but the general practice is to blend at some stage of precipitation or storage at the plant.

Blending at the plants is done in the storage tanks, the "make tanks," and the accumulator tanks, and at times in the coils while the condensate is still in process of precipitation and in contact with the high-pressure gas. Operators using these methods claim definite increases in production for each successively earlier point in the process of cooling and precipitation in the high-pressure units at which blending is accomplished.

Naphthas having an end point of approximately 400° F. are in general use as blending stocks, but at some plants where regular supplies of this stock could not be obtained, distillates having the end points and gravities of kerosene are used.

Some blending companies use with the usual naphthas small quantities of "straight" still-run California gasoline, specific gravity 58° B., and Mid-Continent and eastern grades, specific gravity 66° to 68° B., in order to increase the proportions of those hydrocarbons of which the naphtha and the condensate contain only small percentages.

THE ADVANCEMENT OF THE INDUSTRY.

Since the first commercial gas compression plants were established, about 15 years ago, in the eastern oil fields, marked advancement has been made in the mechanical and commercial phases of the natural-gas gasoline industry.

Up to about five or six years ago, most of the plants consisted of the simplest forms of gas pumps, single-stage compressors, and cooling coils, were operated only on rich casing-head gas that would produce 4 to 6 gallons of condensate, and had a capacity of not more than 200,000 or 300,000 cubic feet daily.

At present plants are in operation treating 6,000,000 to 9,000,000 cubic feet daily of gas yielding as low as 1 gallon of condensate per 1,000 cubic feet, using pressures of 250 and 300 pounds per square inch in two stages of compression, with elaborate systems of cooling the gas with water before compression and after each stage of compression. The water used is cooled below normal temperatures by induced aeration and radiation.

In some plants the gas is further cooled by expanding the dry treated gas through the cylinders of an expansion engine and using the cold expanded gas to cool the high-pressure gas from the water-cooled coils. Temperatures as low as 0° F. are often obtained, causing the precipitation of nearly all the condensable fractions commercially valuable for making gasoline.

FORMULAS AND TABLES GOVERNING THE FLOW OF GAS IN PIPE LINES.

Formulas governing the flow of gas in pipes have been worked out by Weymouth and discussed by him in a paper^a published by the American Society of Mechanical Engineers. These formulas with that part of Weymouth's report that relates to the flow of gas in pipe lines are given in the following:

TRANSMISSION OF NATURAL GAS.

By THOMAS R. WEYMOUTH.

In the design of pipe lines for the transmission of natural gas from the field to the points of consumption it is necessary to make use of a formula expressing the relations to each other of the quantity and initial and final pressures of the gas, and the diameter and length of line. Many such formulas have been proposed giving widely differing results. In nearly all of them the flow is stated as varying as the square root of the fifth power of the pipe diameter, and either the coefficient of friction is considered constant, or a different coefficient is given for each diameter of pipe. This serves well enough where the diameter is known and any one of the other quantities expressed by the formula is desired, but is somewhat awkward when it is desired to ascertain the diameter of line necessary to meet the other given conditions.

The author has derived a new formula which he believes expresses the relationship of the quantities involved even more closely than any heretofore offered. It is based on isothermal flow, and the variation in the value of the coefficient of friction is provided for without complicating the formula, yet permitting the required diameter of line to be ascertained readily.

The expression for the initial velocity of any gas flowing in a pipe is given by Unwin^b as

$$(1) \quad u_1 = \sqrt{\frac{gCTm(P_1^2 - P_2^2)}{fP_1^3}}$$

u_1 =initial velocity, in feet per second.

g =acceleration due to gravity.

C =thermodynamic constant of the flowing gas = $\frac{PV}{T}$

T =absolute temperature of gas.

m =hydraulic mean radius of the pipe = $\frac{D}{4}$.

P_1 =absolute initial pressure of the gas, in pounds per square inch.

P_2 =absolute final pressure of the gas, in pounds per square inch.

f =coefficient of friction.

l =length of line, in feet.

^a Weymouth, T. R., Problems in natural-gas engineering: Trans. Am. Soc. Mech. Eng., vol. 34, 1912, pp. 185-224.

^b Unwin, W. C., Transmission and distribution of power from central stations, 1892, p. 259.

Let

C_a = thermodynamic constant for air.

G = specific gravity of flowing gas, air = 1.0.

D = diameter of pipe, in feet.

d = diameter of pipe, in inches.

$$\text{Then } C = \frac{C_a}{G}, \text{ and } m = \frac{D}{4} = \frac{d}{48}$$

Hence

$$(2) \quad u_1 = \left[\frac{g C_a T (P_1^2 - P_2^2) d}{48 G f P_1^2} \right]^{\frac{1}{2}}$$

If q = quantity of gas flowing per second, based on absolute pressure and temperature of P_o and T_o ,

$$A = \text{area of cross section of pipe in square feet} = \frac{\pi d^2}{4 \times 144}$$

Then

$$(3) \quad q = u_1 A \frac{P_1 T_o}{P_o T} = u_1 \left(\frac{\pi d^2}{4 \times 144} \right) \frac{T_o P_1}{P_o T} \left(\frac{\pi}{576} \right) \frac{T_o}{P_o} \left[\frac{g C_a (P_1^2 - P_2^2) d^5}{48 G T f l} \right]^{\frac{1}{2}}$$

If

Q = flow in cubic feet per hour, based on P_o and T_o , and

L = length of line, in miles,

Then

$$l = 5280 L$$

and

$$Q = 3,600 q$$

$$(4) \quad Q = \frac{3,600 \pi}{576 \sqrt{48} \times 5280} \frac{T_o}{P_o} \sqrt{\frac{(P_1^2 - P_2^2) d^5}{G T f L}} \left[\frac{(P_1^2 - P_2^2) d^5}{G T f L} \right]^{\frac{1}{2}}$$

Taking $g = 32.17$ and $C_a = 53.33$,

$$(5) \quad Q = 1.6156 \frac{T_o}{P_o} \left[\frac{(P_1^2 - P_2^2) d^5}{G T f L} \right]^{\frac{1}{2}}$$

Experiments on the flow of air in pipes of different diameters indicate that the coefficient of friction f is a variable, decreasing with increasing diameters of line. A great many such experiments have been collected and published in "Compressed Air," by Elmo G. Harris, from which, by the use of equation 5, the coefficients of friction have been computed and plotted in figure 14.

In the reports of these tests no statements were made as to the method of measuring the quantity of gas flowing, and it is quite probable that many of the results are inaccurate in this respect. Notwithstanding this, however, the nature of the variation of f with the diameter is evident, and the curve represented by the equation

$$f = \frac{0.008}{\sqrt[3]{d}}$$

gives a fair average of the loci of the points plotted. Inserting this value of f in equation 5, the expression becomes

$$(6) \quad Q = 18.062 \frac{T_o}{P_o} \left[\frac{(P_1^2 - P_2^2) d^{5\frac{1}{3}}}{G T L} \right]^{\frac{1}{2}}$$

Equation 6 is the general formula for the flow of gas in long pipe lines.

In 1901 Forrest M. Towl conducted an extended test on an 8-inch line, 70 miles long, supplying gas to Buffalo, the results of which were published in a bulletin issued by Columbia University in 1911. Previous to the test the line had been repaired and tested for leaks, and was known to be practically gas-tight. The flow was measured by standardized Pitot tubes, which gave results accurate within less than 1 per cent. The specific gravity G of the flowing gas was 0.64, its temperature

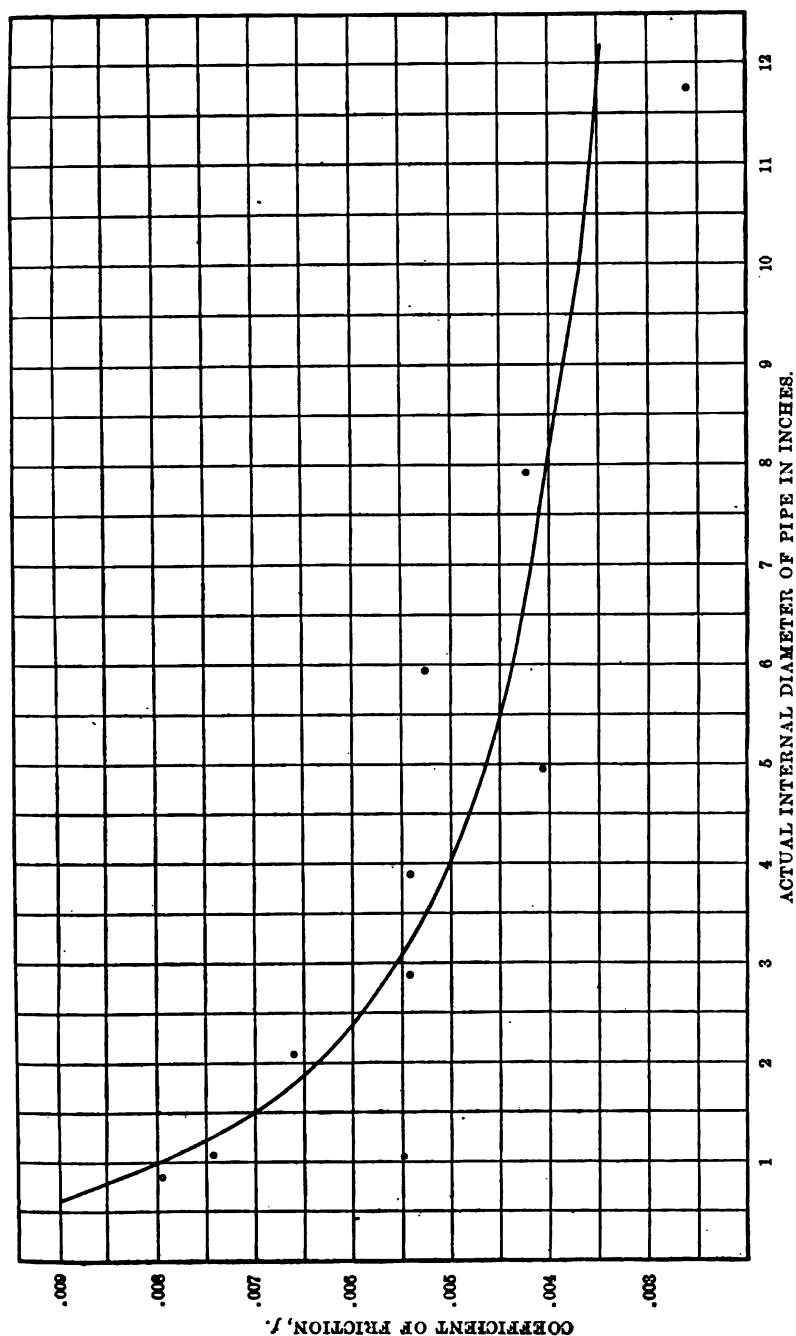


FIGURE 14.—Curve for coefficient of friction for flow of gas in pipes. Coefficient of friction f equals $\frac{0.008}{d^{1/4}}$, where d is internal diameter of pipe in inches.

32° F., or $T=492^{\circ}$ absolute. The temperature basis on which the gas was measured was 50° F., or $T_g=510^{\circ}$ absolute, and the pressure basis was 4 ounces above 14.4 pounds, or $P_g=14.65$ pounds per square inch absolute. In a length of pipe 70.32 miles long P_1 and P_2 were 210 and 41 pounds per square inch absolute, respectively. The actual diameter of the pipe was 7.981 inches, and the rate of flow by Pitot tube was found to be 221,000 cubic feet per hour.

Inserting these quantities in formula 1 and solving for flow, it becomes

$$Q=221,400 \text{ cubic feet per hour,}$$

or less than 0.2 per cent greater than the actual flow as measured.

Assuming gas standard conditions of measurement basis, namely, 60° F. and 14.65 pounds absolute pressure, and that the average flowing temperature of the gas throughout the year will be 40° F., the formula becomes

$$(7) \quad Q=28.66 \left[\frac{(P_1^2 - P_2^2) d^{5/4}}{LG} \right]^{1/4}$$

and, if an average specific gravity of 0.60 be assumed,

$$(8) \quad Q=37 \left[\frac{(P_1^2 - P_2^2) d^{5/4}}{L} \right]^{1/4}$$

Formula 8 is of practical use in designing lines for the transmission of natural gas. It is used as given, or in a transposed form, for all problems relating to single lines of uniform diameter.

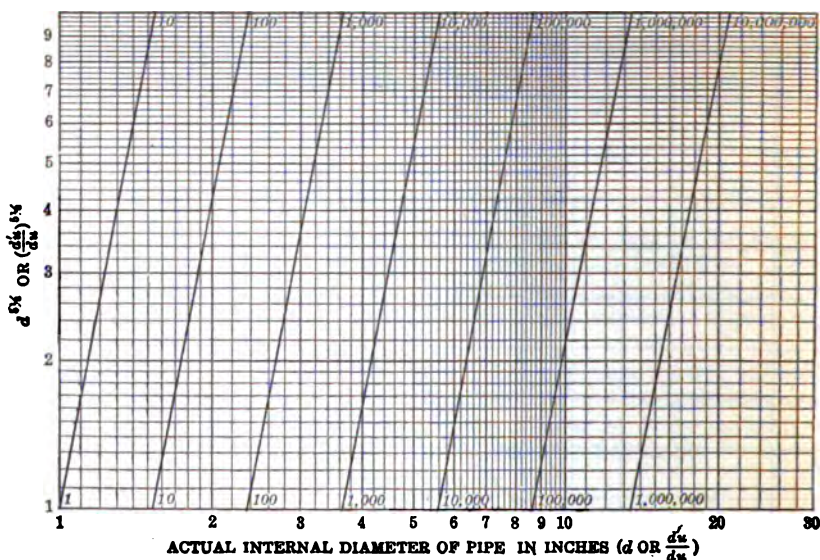


FIGURE 15.—Broken curve showing values of equivalent lengths of different diameters of pipe.

If a line is composed of several lengths, $L_1, L_2, \dots L_n$, of diameters $d_1, d_2, \dots d_n$, each of these lengths must be transformed into an equivalent length of one chosen diameter, by means of the formula

$$(9) \quad L_n^1 = L_n \left[\frac{d_n^1}{d_n} \right]^{5/4}$$

These equivalent lengths added together will give $L=L_1, L_2, \dots L_n$, which is the value for L in formula 8.

Values of equivalent lengths for different diameters can be most conveniently ascertained by the use of the curves in figure 15, which consist of plots of the values of $d^{5/4}$ for varying values of d . Values taken from these curves are convenient to use directly in the pipe-line formula, equation 8, whereas they are most simply used in equation 9 as values of the $5\frac{1}{4}$ power of the diameter ratios.

TABLES.

VOLUME OF FLOW FROM DIFFERENT SIZES OF PIPE AT VARIOUS PRESSURES.

The number of cubic feet of gas of a specific gravity of 0.6 (air equaling 1.0) that will flow from the mouth of a 1-inch pipe in 24 hours is given in Table 10 following. The pressure of the container is taken as 4 ounces above an assumed atmospheric pressure of 14.4 pounds to the square inch, and the temperature of the flowing gas and the container is assumed to be 60° F. If the diameter of the pipe is other than 1 inch, multiply the discharge value given in Table 10 by the square of the actual diameter of the pipe, as found in Table 11.

TABLE 10.—Rate of flow of natural gas in pipe 1 inch in inside diameter at various pressures.^a

Observed gage pressure.			Flow, cubic feet per day.	Observed gage pressure.			Flow, cubic feet per day.
Inches of mercury.	Inches of water.	Pounds per square inch.		Inches of mercury.	Inches of water.	Pounds per square inch.	
.....	0.1	0.0036	12,390	10.17		5.0	436,200
.....	0.2	0.0073	17,560	11.18		5.5	456,200
.....	0.3	0.0109	21,480	12.20		6.0	473,750
.....	0.5	0.0182	27,720	13.21		6.5	489,840
0.05	0.7	0.0254	32,820	14.23		7.0	505,920
0.7	1.0	0.0374	39,210	15.25		7.5	522,010
0.11	1.5	0.0545	48,030	16.26		8.0	538,500
0.15	2.0	0.0727	55,340	18.30		9.0	565,970
0.22	3.0	0.109	67,910	20.33		10.0	589,270
0.29	4.0	0.145	78,410	24.39		12.0	633,340
0.37	5.0	0.182	87,670	28.46		14.0	675,000
0.52	7.0	0.254	103,600	32.53		16.0	713,550
0.74	10.0	0.3636	123,000	36.60		18.0	748,650
1.02	13.75	0.50	146,220	40.66		20.0	779,350
1.52	20.62	0.75	175,350	50.81		25.0	845,150
2.03	27.5	1.00	201,800	61.00		30.0	902,180
3.05	41.25	1.5	247,840	71.16		35.0	954,820
4.07	55.0	2.0	285,130			40.0	999,680
5.08	68.75	2.5	316,500			45.0	1,036,700
6.10	82.50	3.0	344,350			50.0	1,072,000
7.12	96.25	3.5	370,000			55.0	1,106,880
8.13	110.0	4.0	393,000			60.0	1,137,600
8.15		4.5	415,270				

^a Thompson, A. B., Oil-field development and petroleum mining, Lind., 1916, pp. 578-579; quoted by Johnson, R. H., and Huntley, L. G., Oil and gas production, 1916, p. 9.

In correcting for temperature of flowing gas, where observed, of 30°, 40°, 50°, and 60° F., add 4, 3, 2, and 1 per cent, respectively. To change the result, as found by this table, to that for any other specific

gravity of gas than 0.6, multiply by $\sqrt{\frac{0.6}{\text{specific gravity of gas}}}$

TABLE 11.—*Multipliers for pipe of diameters other than 1 inch.^a*

Diam- eter of opening.	Multi- plier.	Diam- eter of opening.	Multi- plier.	Diam- eter of opening.	Multi- plier.	Diam- eter of opening.	Multi- plier.	Diam- eter of opening.	Multi- plier.
<i>Inch.</i>		<i>Inches.</i>		<i>Inches.</i>		<i>Inches.</i>		<i>Inches.</i>	
1	0.0038	1	1.00	4	16.00	6	36.00	8	64.00
1½	0.0156	1½	2.25	4½	18.00	6½	39.00	8½	68.00
2	0.0625	2	4.00	5	25.00	6½	43.00	9	81.00
2½	0.2500	2½	6.25	5½	26.90	7	49.00	10	100.00
3	0.5625	3	9.00	5½	31.60	7½	52.50	12	144.00

^a Johnson, R. H., and Huntley, L. G., Oil and gas production, 1916, p. 355.TABLE 12.—*Variation in volume of 100 cubic feet (100 per cent) of gas at constant temperature under various gage pressures.^b*

Pressure per square inch.	Volume.	Pressure per square inch.	Volume.	Pressure per square inch.	Volume.
<i>Ounces.</i>	<i>Per cent.</i>	<i>Pounds.</i>	<i>Per cent.</i>	<i>Pounds.</i>	<i>Per cent.</i>
0	100.0	4	78.6	2½	42.3
2	99.1	5	74.6	30	32.8
4	98.3	6	71.0	40	27.8
6	97.5	7	67.7	50	22.7
8	96.7	8	64.7	75	16.8
10	95.9	9	62.0	100	12.8
12	95.1	10	59.5	150	8.9
14	94.3	12	55.0	200	6.8
<i>Pounds.</i>					
1	93.6	14	51.5	250	5.5
2	88.0	16	47.8	300	4.6
3	83.0	18	44.9	400	3.5

^b Johnson, R. H., and Huntley, L. G., Principles of oil and gas production, 1916, p. 355.

CHANGE IN VOLUME OF GAS WITH CHANGE IN TEMPERATURE.

In Table 13 following, the standard is taken at 60° F. and 14.4 inches of mercury plus 0.25=14.65 inches of mercury. Absolute zero=460° F. below freezing=488° below 60° F. The specific gravity of the natural gas is taken at 0.6, air being 1. The same 1,000 cubic feet of gas at 60° F. will measure 1,041 cubic feet at 80° and 959 cubic feet at 40°. The percentage of the decrease and increase below or above 60° F.; the specific gravity of the gas at temperatures below and above 60° F.; also weight of 1,000 cubic feet of gas and air at the different temperatures is shown. For each degree there is a change of 0.002056 in volume.

TABLE 13.—*Change in volume of 1,000 feet of air or natural gas, owing to change in temperature.^a*

Tempera- ture.	Volume of 1,000 cubic feet of gas measured at tem- peratures other than 60° F.	Loss or gain in volume.	Specific gravity (specific gravity— 0.6 at 60° F.).	Weight of 1,000 cubic feet of gas (0.6 specific gravity at 60° F.).	Weight of 1,000 cubic feet of air.
[°] F.	Cubic feet.	Per cent.		Pounds.	Pounds.
0	877	—12.3	0.6841	58.82	85.97
10	897	—10.3	0.6689	56.41	84.33
20	918	—8.2	0.6536	54.04	82.69
32	943	—5.7	0.6362	51.36	80.73
40	959	—4.1	0.6256	49.68	79.43
50	980	—2.0	0.6124	47.63	77.77
60	1,000	0.0	0.6000	45.67	76.12
70	1,020	+2.0	0.5879	43.78	74.48
80	1,041	+4.1	0.5763	41.96	72.83
90	1,061	+6.1	0.5652	40.23	71.10
100	1,082	+8.2	0.5545	38.56	69.55
110	1,102	+10.2	0.5442	36.95	67.90
120	1,122	+12.3	0.5343	35.40	66.26
130	1,143	+14.3	0.5247	34.10	64.62
140	1,163	+16.3	0.5157	32.47	62.98
150	1,184	+18.4	0.5067	31.07	61.33
160	1,204	+20.4	0.4981	29.72	59.69
170	1,225	+22.5	0.4898	28.42	58.05
180	1,245	+24.5	0.4818	27.17	56.40
190	1,265	+26.6	0.4739	25.94	54.76
200	1,285	+28.6	0.4665	24.78	53.12
210	1,306	+30.7	0.4591	23.63	51.48
212	1,311	+31.1	0.4576	23.41	51.16

^a Westcott, H. P., Handbook of natural gas, Erie, Pa., 1913, p. 379.

SPECIFIC GRAVITY AND BAUMÉ SCALE COMPARED.

Table 14 shows Baumé hydrometer readings from 10° to 90° B. with corresponding specific gravity, and also the corresponding weight of gasoline in pounds per United States gallon at 60° F.

TABLE 14.—*Baumé scale and specific gravity equivalents.*^a

° B.	Specific gravity.	Pounds in gallon.	° B.	Specific gravity.	Pounds in gallon.	° B.	Specific gravity.	Pounds in gallon.
10	1.000	8.33	37	0.8383	6.99	64	0.7216	6.01
11	0.9929	8.27	38	0.8333	6.94	65	0.7179	5.98
12	0.9859	8.21	39	0.8284	6.90	66	0.7143	5.96
13	0.9790	8.15	40	0.8235	6.86	67	0.7107	5.92
14	0.9722	8.10	41	0.8187	6.82	68	0.7071	5.89
15	0.9655	8.04	42	0.8140	6.78	69	0.7035	5.86
16	0.9589	7.99	43	0.8092	6.74	70	0.7000	5.83
17	0.9524	7.93	44	0.8046	6.70	71	0.6965	5.80
18	0.9459	7.88	45	0.8000	6.66	72	0.6931	5.77
19	0.9396	7.83	46	0.7955	6.62	73	0.6897	5.74
20	0.9333	7.77	47	0.7910	6.59	74	0.6863	5.71
21	0.9272	7.72	48	0.7865	6.55	75	0.6829	5.69
22	0.9211	7.67	49	0.7821	6.51	76	0.6796	5.66
23	0.9150	7.62	50	0.7778	6.48	77	0.6763	5.63
24	0.9091	7.57	51	0.7735	6.44	78	0.6731	5.60
25	0.9032	7.52	52	0.7692	6.40	79	0.6699	5.58
26	0.8974	7.47	53	0.7650	6.37	80	0.6677	5.55
27	0.8917	7.42	54	0.7609	6.33	81	0.6635	5.52
28	0.8861	7.38	55	0.7568	6.30	82	0.6604	5.50
29	0.8805	7.33	56	0.7527	6.27	83	0.6573	5.47
30	0.8750	7.29	57	0.7487	6.23	84	0.6542	5.45
31	0.8696	7.24	58	0.7447	6.20	85	0.6512	5.42
32	0.8642	7.20	59	0.7407	6.17	86	0.6482	5.40
33	0.8589	7.15	60	0.7368	6.13	87	0.6452	5.37
34	0.8537	7.11	61	0.7330	6.10	88	0.6422	5.35
35	0.8485	7.07	62	0.7292	6.07	89	0.6393	5.32
36	0.8434	7.02	63	0.7254	6.04	90	0.6364	5.30

^a U. S. Bureau of Standards, United States standard tables for petroleum oils, Circular 57, 1916, p. 57.

NOTE.—Degrees Baumé may be converted to specific gravity by adding 130 to the number of degrees Baumé and dividing the sum by 140.

CAPACITIES OF ORIFICES.

Table 15 shows the capacities of orifices, for testing small flows of natural gas, ranging from one-eighth of an inch to $1\frac{1}{4}$ inches in diameter in plates one-eighth of an inch thick.

TABLE 15.—Capacities of orifices for testing flows of natural gas from small gas wells and casing-head gas from oil wells. a

[Temperature, 60° F.; atmospheric pressure, 14.4 pounds per square inch.]

THREE-EIGHTS-INCH ORIFICE IN PLATE ONE-EIGHTH INCH THICK.

Pressure.	Capacity, in cubic feet per 24 hours, at specific gravity of—											
	0.6	0.65	0.7	0.75	0.8	0.85	0.9	0.95	1	1.05	1.10	1.15
<i>Inches of water.</i>												
0.5.....	2,370	2,180	2,100	2,030	1,970	1,910	1,850	1,810	1,760	1,720	1,680	1,640
1.0.....	3,460	3,230	3,200	3,090	3,000	2,910	2,820	2,750	2,690	2,620	2,550	2,480
1.5.....	4,310	4,140	4,090	3,960	3,780	3,620	3,480	3,330	3,180	3,020	2,850	2,690
2.0.....	4,830	4,640	4,470	4,320	4,180	4,040	3,900	3,740	3,580	3,410	3,230	3,060
2.5.....	5,400	5,190	5,000	4,830	4,680	4,540	4,410	4,260	4,100	3,930	3,750	3,570
3.0.....	5,770	5,550	5,350	5,170	5,000	4,850	4,720	4,580	4,420	4,250	4,070	3,890
3.5.....	6,230	6,050	5,830	5,630	5,450	5,290	5,140	4,990	4,820	4,650	4,470	4,290
4.0.....	6,650	6,390	6,160	5,950	5,760	5,590	5,430	5,280	5,110	4,940	4,760	4,580
4.5.....	7,100	6,830	6,590	6,370	6,160	5,980	5,800	5,630	5,450	5,270	5,090	4,910
5.0.....	7,600	7,330	7,110	6,870	6,650	6,450	6,270	6,100	5,920	5,740	5,560	5,380
5.5.....	8,100	7,790	7,500	7,250	7,020	6,810	6,620	6,440	6,270	6,100	5,920	5,740
6.0.....	8,530	7,970	7,680	7,420	7,180	6,970	6,770	6,590	6,420	6,250	6,080	5,910

ONE-HALF-INCH ORIFICE IN PLATE ONE-EIGHTH INCH THICK.

0.5.....	4,490	4,320	4,160	4,020	3,890	3,770	3,670	3,570	3,480	3,400	3,320	3,250
1.0.....	6,260	6,010	5,790	5,600	5,440	5,290	5,110	4,970	4,850	4,730	4,620	4,520
1.5.....	7,900	7,590	7,310	7,070	6,860	6,640	6,450	6,280	6,120	5,970	5,830	5,690
2.0.....	9,140	8,790	8,460	8,170	7,910	7,680	7,460	7,260	7,080	6,910	6,750	6,600
2.5.....	9,900	9,520	9,170	8,850	8,550	8,290	8,050	7,830	7,620	7,430	7,250	7,080
3.0.....	11,150	10,720	10,330	9,960	9,600	9,270	8,960	8,640	8,350	8,060	7,790	7,530
3.5.....	12,020	11,550	11,130	10,750	10,410	10,100	9,810	9,550	9,310	9,080	8,860	8,650
4.0.....	12,900	12,390	11,950	11,440	11,060	10,720	10,400	10,100	9,810	9,540	9,290	9,050
4.5.....	13,480	12,950	12,480	12,050	11,670	11,320	11,000	10,710	10,440	10,190	9,950	9,730
5.0.....	14,130	13,570	13,080	12,640	12,230	11,870	11,530	11,210	10,940	10,680	10,430	10,200
5.5.....	14,690	14,110	13,600	13,130	12,720	12,340	11,990	11,670	11,380	11,100	10,850	10,610
6.0.....	15,210	14,620	14,080	13,610	13,170	12,780	12,420	12,080	11,780	11,500	11,230	10,980

a Wescott, H. P., Handbook of casing-head gas, 1916, pp. 86-82.

TABLE 15.—Capacities of orifices for testing flows of natural gas from small gas wells and casing-head gas from oil wells—Continued.
THREE-FOURTHS-INCH ORIFICE IN PLATE ONE-EIGHTH INCH THICK.

Pressure.	Capacity, in cubic feet per 24 hours, at specific gravity of—													
	0.6	0.65	0.7	0.75	0.8	0.85	0.9	0.95	1	1.05	1.10	1.15	1.20	1.30
<i>Inches of water.</i>														
0.5.....	10,560	10,150	9,780	9,450	9,150	8,880	8,630	8,400	8,180	7,990	7,800	7,630	7,470	7,170
1.0.....	14,530	13,960	13,450	12,980	12,560	12,210	11,860	11,550	11,260	10,960	10,730	10,500	10,280	9,870
1.5.....	17,720	17,030	16,410	15,850	15,350	14,910	14,470	14,060	13,700	13,400	13,080	12,800	12,530	12,040
2.0.....	20,390	19,590	18,870	18,230	17,650	17,130	16,650	16,200	15,780	15,410	15,060	14,730	14,420	13,850
2.5.....	22,740	21,850	20,950	20,340	19,760	19,210	18,710	18,240	17,800	17,400	17,030	16,680	16,350	15,700
3.0.....	24,880	23,900	22,930	22,250	21,590	20,940	20,310	19,710	19,140	18,610	18,100	17,600	17,120	16,380
3.5.....	26,990	25,930	24,980	24,140	23,370	22,670	22,030	21,450	20,900	20,400	19,930	19,490	19,060	18,330
4.0.....	28,970	27,830	26,920	26,020	25,090	24,340	23,650	23,020	22,440	21,900	21,400	20,920	20,480	19,740
4.5.....	30,800	29,590	28,510	27,560	26,570	25,870	25,150	24,470	23,960	23,480	23,020	22,580	22,160	21,400
5.0.....	32,500	31,230	30,000	28,970	27,850	26,910	26,150	25,430	24,900	24,570	24,260	23,960	23,680	22,890
5.5.....	34,080	32,740	31,530	30,480	29,310	28,330	27,390	26,680	26,140	25,760	25,400	25,050	24,720	23,900
6.0.....	35,630	34,230	32,990	31,870	30,660	29,640	28,660	27,820	27,200	26,830	26,480	26,140	25,820	24,970
1.....	26,440	25,440	24,500	23,600	22,720	21,880	21,060	20,280	19,540	18,840	18,180	17,560	16,980	16,340
2.....	37,510	36,040	34,750	33,500	32,280	31,100	30,040	29,000	28,000	27,040	26,120	25,220	24,360	23,540
3.....	46,640	44,640	42,800	41,000	39,240	37,520	35,840	34,200	32,600	31,040	29,520	28,040	26,600	25,200
4.....	52,630	50,330	48,100	45,900	43,740	41,620	39,540	37,500	35,500	33,540	31,620	29,740	27,900	26,100
5.....	57,890	55,330	52,900	50,500	48,140	45,820	43,540	41,300	39,100	36,940	34,820	32,740	30,700	28,700
6.....	62,140	59,470	56,900	54,400	51,940	49,520	47,140	44,800	42,500	40,240	38,020	35,840	33,700	31,600
7.....	65,110	62,470	59,900	57,400	54,940	52,520	50,140	47,800	45,500	43,240	41,020	38,840	36,700	34,600
8.....	67,950	65,330	62,800	60,300	57,840	55,420	53,040	50,700	48,400	46,140	43,920	41,740	39,600	37,500
9.....	70,680	68,000	65,500	63,000	60,540	58,120	55,740	53,400	51,100	48,840	46,620	44,440	42,300	40,200
10.....	73,340	70,600	68,100	65,600	63,140	60,720	58,340	56,000	53,700	51,440	49,220	47,040	44,900	42,800
11.....	75,950	73,160	70,600	68,100	65,600	63,140	60,720	58,340	56,000	53,700	51,440	49,220	47,040	44,900
12.....	78,520	75,680	73,000	70,400	67,800	65,200	62,600	60,000	57,400	54,800	52,200	49,600	47,000	44,400

1-INCH ORIFICE IN PLATE ONE-EIGHTH INCH THICK.

1/2-INCH ORIFICE IN PLATE ONE-EIGHTH INCH THICK.

Inches of mercury.															
0.5	39,900	41,100	42,500	44,000	45,700	47,500	49,000	50,800	52,100	53,400	54,900	56,500	58,200	59,900	61,600
1.0	61,600	63,300	65,000	66,800	68,600	70,400	72,200	74,000	75,800	77,600	79,400	81,200	83,000	84,800	86,600
1.5	86,600	88,400	90,200	92,000	93,800	95,600	97,400	99,200	101,000	102,800	104,600	106,400	108,200	110,000	111,800
2.0	111,800	113,600	115,400	117,200	119,000	120,800	122,600	124,400	126,200	128,000	129,800	131,600	133,400	135,200	137,000
2.5	137,000	138,800	140,600	142,400	144,200	146,000	147,800	149,600	151,400	153,200	155,000	156,800	158,600	160,400	162,200
3.0	162,200	164,000	165,800	167,600	169,400	171,200	173,000	174,800	176,600	178,400	180,200	182,000	183,800	185,600	187,400
3.5	187,400	189,200	191,000	192,800	194,600	196,400	198,200	200,000	201,800	203,600	205,400	207,200	209,000	210,800	212,600
4.0	212,600	214,400	216,200	218,000	219,800	221,600	223,400	225,200	227,000	228,800	230,600	232,400	234,200	236,000	237,800
4.5	237,800	239,600	241,400	243,200	245,000	246,800	248,600	250,400	252,200	254,000	255,800	257,600	259,400	261,200	263,000
5.0	263,000	264,800	266,600	268,400	270,200	272,000	273,800	275,600	277,400	279,200	281,000	282,800	284,600	286,400	288,200
5.5	288,200	290,000	291,800	293,600	295,400	297,200	299,000	300,800	302,600	304,400	306,200	308,000	309,800	311,600	313,400
Inches of water.															
0.5	32,780	31,490	30,350	29,220	28,390	27,540	26,760	26,050	25,390	24,779	24,210	23,690	23,180	22,270	21,458
1.0	46,250	44,440	42,830	41,380	40,000	38,800	37,770	36,760	35,830	34,968	34,160	33,410	32,710	31,420	30,280
1.5	56,600	54,380	52,410	50,630	49,020	47,560	46,210	44,980	43,850	42,790	41,810	40,890	40,000	38,440	36,920
2.0	64,840	62,300	60,040	58,000	56,160	54,480	52,940	51,530	50,230	49,020	47,880	46,840	45,860	44,050	42,450
2.5	72,400	69,570	67,040	64,760	62,710	60,830	59,120	57,540	56,090	54,735	53,480	52,300	51,200	49,140	47,400
3.0	77,950	74,920	72,200	69,750	67,540	65,520	63,670	61,970	60,410	58,950	57,600	56,330	55,150	52,980	51,050
3.5	84,490	81,180	78,220	75,570	73,170	70,980	68,980	67,140	65,430	63,870	62,400	61,050	59,780	57,400	55,310
4.0	91,400	87,810	84,600	81,750	79,150	76,780	74,620	72,630	70,800	69,090	67,500	66,020	64,630	62,040	59,840
4.5	97,810	93,960	90,540	87,490	84,710	82,180	79,870	77,730	75,710	73,940	72,280	70,690	69,170	66,450	64,040
5.0	103,260	99,210	95,610	92,370	89,430	86,770	84,310	82,060	80,000	78,070	76,280	74,600	73,000	70,100	67,610
5.5	107,230	103,000	99,280	95,910	92,870	90,080	87,550	85,210	83,060	81,060	79,194	77,460	75,820	72,850	70,200
6.0	110,700	106,300	102,300	98,700	95,400	92,300	89,350	86,550	83,890	81,370	78,980	76,700	74,510	71,400	68,550
6.5	113,700	109,100	105,000	101,300	97,900	94,700	91,650	88,750	85,900	83,190	80,620	78,180	75,860	72,600	69,500
7.0	116,300	111,500	107,300	103,500	100,000	96,700	93,550	90,550	87,600	84,790	82,120	79,580	77,160	73,800	70,550
7.5	118,600	113,600	109,300	105,400	101,800	98,400	95,150	92,050	89,000	86,090	83,320	80,680	78,160	74,750	71,400
8.0	120,700	115,500	111,100	107,100	103,400	100,000	96,700	93,500	90,350	87,250	84,290	81,460	78,750	75,150	71,600
8.5	122,600	117,200	112,700	108,600	104,800	101,300	97,950	94,700	91,450	88,250	85,100	82,000	78,950	75,850	72,700
9.0	124,400	118,800	114,200	110,000	106,100	102,500	99,050	95,700	92,350	89,000	85,750	82,500	79,350	76,100	72,800
9.5	126,000	120,300	115,600	111,300	107,400	103,800	100,300	96,850	93,400	90,000	86,600	83,200	79,800	76,350	72,900
10.0	127,500	121,700	116,900	112,500	108,500	104,900	101,300	97,800	94,250	90,750	87,250	83,750	80,250	76,700	73,100
10.5	128,900	123,000	118,100	113,600	109,500	105,900	102,300	98,750	95,150	91,600	88,050	84,500	80,950	77,300	73,650
11.0	130,200	124,200	119,200	114,600	110,400	106,700	103,000	99,350	95,650	92,000	88,350	84,650	80,950	77,200	73,450
11.5	131,500	125,400	120,300	115,600	111,300	107,500	103,700	100,000	96,250	92,500	88,750	84,950	81,150	77,300	73,450
12.0	132,700	126,500	121,300	116,500	112,100	108,300	104,500	100,700	96,900	93,050	89,200	85,350	81,500	77,600	73,650

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Bulletin 152

Law Serial 11

DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

ABSTRACTS OF CURRENT DECISIONS

ON

MINES AND MINING

REPORTED FROM

JANUARY TO APRIL, 1917

BY

J. W. THOMPSON



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ABSTRACTS OF CURRENT DECISIONS ON MINES AND MINING, JANUARY TO APRIL, 1917.

By J. W. THOMPSON.

MINERALS AND MINERAL LANDS.

MINERALS.

PROCESS OF ORE CONCENTRATION—OIL FLOTATION—INFRINGEMENT OF PATENT.

The Sullman, Picard & Ballot patent, No. 835120, is a process of concentrating ore which consists in mixing the powdered ore with water, adding a small proportion, a fraction of 1 per cent on the ore, of an oily liquid having a preferential affinity for metal-liferous matter, agitating the mixture until the oil-coated mineral matter forms into a froth and separating the froth from the remainder by flotation. The process is known as an air-flotation process which consists in the use of a frothing agent in conjunction with such agitation of the ore pulp as will distribute the metallic particles of the ore throughout the mixture and produce bubbles of air and bring them in contact in the mixture with the metallic particles so distributed. The bubbles will become attached to such metallic particles, carrying them separately from the particles of gangue up through the surface of the mixture where they can be readily collected by skimming, by overflow, or by the use of other well-known devices. The air-flotation process was a patentable invention in the discovery made in March, 1905, and was not anticipated and is valid. Prior to that time there had been no suggestion in the arts that a proportion of 0.1 per cent of oil to ore or of any other fraction of 1 per cent of oil to ore would or might result in successful concentration. While the ascertainment that such a minute proportion of oil would affect a successful concentration of ore through a flotation process was a discovery, it was nevertheless of such a character, viewed with respect to the circumstances under which it was made, as to involve invention and confer patentability.

Minerals Separation v. Miami Copper Co., 237 Fed., 609, pp. 611, 616.

ORE CONCENTRATION PROCESS—VALIDITY OF PATENT.

Claims 1 and 12 of the Sullman, Picard & Ballot patent, No. 835120, for a process of concentrating ore, known as the air-flotation process, are valid; but claim 9 is so indefinite as to render the claim void. The object of the Sullman, Greenway & Higgins patent, No. 962678, dated June 28, 1910, for improvements in ore concentration, is to separate metalliferous matter, graphite, and the like from gangue by means of oil, fatty acid, or other substances which have a preferential affinity for metalliferous matter over gangue, and it is valid. The Greenway patent, No. 1099699, dated June 9, 1914, for improvements in the concentration of ores is invalid, as in view of the prior art the omission to use acid in the process can not confer patentability.

Minerals Separation v. Miami Copper Co., 237 Fed., 609, pp. 630-634-636.

SULLMAN, PICARD & BALLOT PATENT—VALIDITY.

The process of the Sullman, Picard & Ballot patent, dated November 6, 1906 (No. 835120), consists in the use of an amount of oil which is "critical" and minute compared with the amount used in prior processes. The amount of oil used in this patent amounts to a fraction of 1 per cent on the ore and in so impregnating with air the mass of ore and water used, by agitation or by beating the air into the mass and causing to rise to the surface of the mass of pulp a froth peculiarly coherent and persistent in character and that is composed of air bubbles with only a trace of oil in them and which carry in mechanical suspension a very high percentage of the metal and metalliferous particles of ore that were contained in the mass of crushed ore subject to the treatment. This froth can be removed and the metal recovered by processes aside from the patent. The process of this patent is not of the metal sinking class and while it may be described as a surface flotation process yet it differs so essentially from all prior processes, in its character, in its simplicity of operation, and in the resulting concentrate, as to constitute a new and patentable discovery. The prior processes required the use of so much oil that they were too expensive to be used on lean ores, to which they were intended to have their chief application. This patent differs essentially from all previous results. Oil, while one of the substances still used, is used in quantities much smaller than in any previous patent and produces a result never before obtained. The minerals are obtained in a froth of peculiar character consisting of air bubbles, which in their covering film have the minerals embedded in such manner that they form a complete surface all over the bubbles. Although the light and easily destructible air bubbles

are covered with a heavy mineral, the froth is stable and different from any froth known before, being so permanent in character that it will stand for 24 hours without any change having taken place. The simplicity of the process consists in adding a minute quantity of oil to the pulp to which acid may or may not be added, and then agitating $2\frac{1}{2}$ to 10 minutes and after a few seconds collecting from the surface the froth which will contain a large percentage of the minerals present in the ore or pulp.

Minerals Separation v. Hyde, 242 U. S., 261, p. 268.

See *Minerals Separation v. British Ore Concentration Syndicate*, 27 R. P. C., 33.

PATENT—OIL-FLOTATION PROCESS.

The processes of oil flotation are divided into two classes as recognized in the patents: The process in the patents of the first class is called the surface flotation process and depends for its usefulness on the oil used being sufficient to collect and hold in mechanical suspension the small particles of metal and metalliferous compounds, and by its buoyancy to carry them to the surface of the mixture of ore, water, and oil, thus making it possible by methods familiar to those skilled in the art to float off the concentrate thus formed into a desired receptacle. The waste material or gangue not being affected by the oil and being heavier than water sinks to the bottom of the containing vessel and is disposed of. The process of the other class is called the metal sinking process, and reverses the action of the surface flotation process in which oil is used to the extent of 4 to 10 per cent of the weight of the metalliferous matter, depending on the character of the ore, for the purpose of agglomerating the oil-coated concentrate in granules heavier than water and these then sink to the bottom of the containing vessel and permit the gangue to be carried away by an upward flowing stream of water.

Minerals Separation v. Hyde, 242 U. S., 261, p. 264.

PATENT—TREATMENT OF ORE.

A number of patents have been granted in this and other countries for processes aiming to make practical use of the property of oil and of oil mixed with acid in the treatment of ores, all of which consists of mixing finely crushed or powdered ore with water and oil and sometimes with acid added, and then in variously treating the mass or pulp thus formed so as to separate the oil when it becomes impregnated or loaded with the metal and metalliferous bearing particles from the valueless gangue, and from the resulting concentrate the minerals are recovered in various ways.

Minerals Separation v. Hyde, 242 U. S., 261, p. 264.

PATENT—AFFINITY OF OIL AND METALS.

In the concentration of ores it has become a well-known fact that oil and oily substances have a selective affinity or attraction for and will unite mechanically with minute particles and metalliferous compounds found in crushed or powdered ores, and will not so unite with the quartz, or rocky nonmetalliferous mineral called gangue. It is also well known that this selective property of oils and oily substances is increased when applied to some ores by the addition of a small amount of acid to the ore and water used in the process of concentration. These principles were the bases of the Haynes, British patent (1860), and United States patents, Everson (1886), Robson (1897), Elmore (1901), and Cattermole, (1904).

Minerals Separation v. Hyde, 242 U. S., 261, p. 264.

SULLMAN, PICARD & BALLOT PATENT—TREATMENT OF DIFFERENT ORES.

The Sullman, Picard & Ballot patent, No. 835120, is not invalid on the ground that when different ores are treated preliminary tests must be made to determine the amount of oil and extent of agitation necessary in order to obtain the best results. Such variation of treatment must be within the scope of the claims, and the certainty which the law requires in patents is not greater than is reasonable, having regard to their subject matter. The composition of ore varies infinitely, each one presenting its special problem, and it is impossible to specify in a patent the precise treatment which would be most successful and economical in each case. The process deals with a large class of substances and the range of treatment with the terms of the claims, while leaving something to the skill of the persons applying the patent, is clearly and sufficiently definite to guide those skilled in the art to its successful application.

Minerals Separation v. Hyde, 242 U. S., 261, p. 270.

SULLMAN, PICARD & BALLOT PATENT—CLAIMS VALID AND INVALID.

The Sullman, Picard & Ballot patent, No. 835120, for a process of oil concentration is valid as to claims 1, 2, 3, 5, 6, 7, 12, and invalid as to claims 9, 10, 11. The validity of claims 4, 8, and 13 was not involved and, therefore, not considered.

Minerals Separation v. Hyde, 242 U. S., 261, p. 271.

ROYALTIES.

Royalties received by a lessor under mineral leases constitute rents and profits of land.

Pettit's Estate, In re, 161 Northwestern, 158, p. 161.

SALE AND CONVEYANCE.**CONSTRUCTION OF OPTION CONTRACT—PAROL EVIDENCE TO EXPLAIN.**

The owners of a large tract of mineral land agreed with certain persons on the sale of the land at a stated price, if such second persons succeeded in making a sale. The persons acting as agents, to facilitate the sale, organized a corporation, and on their request the owners conveyed the land to the corporation, under an agreement that the corporation would reconvey the property to the owners, and notes were executed by the corporation, representing the price agreed upon. It was not the intention of any of the parties to the transaction that if the land was not sold that the corporation should own and hold the land, and be indebted for the amount of the notes. Under such circumstances the deed to the corporation, in the light of the testimony as to the intention of the parties, was nothing more than an option agreement, and on failure of the original parties undertaking to sell the land to make a sale, the deed to the corporation became ineffective, transferred no title, and did not operate as a mortgage to secure the payment of the contract price.

Arizona Copper Estate v. Watts, 237 Fed., 585.

OPTION AGREEMENT TO REPURCHASE—SPECIFIC PERFORMANCE.

A mining company sold and conveyed a tract of land owned by it from part of which it had mined and removed the coal and reserved, by the terms of the deed, the remaining coal under the tract conveyed. The deed also contained an option agreement to repurchase, by the grantor, all or any portion of the premises conveyed for the average price per acre paid by the purchaser. No time was specified in which the repurchase should be made. Specific performance of the agreement to repurchase and reconvey will not be decreed after a lapse of more than 10 years, and where it appeared that the purchaser had placed improvements on the land to the value of \$4,000, with the knowledge of the grantor and where no offer was made to pay for the improvements.

Ando v. Western Coal & Mining Co. (Kansas), 162 Pacific, 344, p. 347.

CONSTRUCTION OF CONTRACT.

The owner of mining claims entered into an agreement with a purchaser whereby the owner sold to the purchaser seven mining claims, together with improvements thereon. By the agreement the purchaser was given a choice to pay the sum of \$16,000 in full for the mining claims, or to pay the sum of \$21,000 under different circumstances. It was to be paid in certain stated installments. If the purchaser elected to pay \$21,000, the payments were to be on

different stated installments, \$9,000 of which was to be paid out of one-half of the net proceeds of the mine. The purchaser elected to pay the \$21,000 upon the stated terms, and paid the sum of \$12,000 thereon and obtained the deed, as provided in the contract. Under the terms and conditions of the contract, the seller could not sue and recover, after the expiration of a period of eight years, the remaining \$9,000 due under the agreement, where it appeared that the purchaser had not operated the mine and there had been, in fact, no net proceeds whatever from the mine.

Johnson v. Geddes (Utah), 161 Pacific, 910, p. 911.

ADVERSE POSSESSION—JUDGMENT QUIETING TITLE.

In an action to quiet title to minerals and mineral lands, the judgment of a court of competent jurisdiction settles and determines the title as against a party claiming by adverse possession or otherwise, and a judgment which determines that an adverse claimant is not the owner and forbids him to be such, and enjoins him from using or trespassing upon the land, and concludes his right to hold it adversely; and any adverse possession had or claimed by him before the institution of the suit is extinguished by the judgment. On the rendition of the judgment, it was his duty to give up the possession and to refrain from any other possession of the land or claim to its ownership. Such a judgment breaks the continuity of the possession by the adverse claimant and he can not, in a subsequent suit against the original owner, or his grantee, base a claim of adverse possession where such possession antedates the time of the rendition of such judgment.

Tennis Coal Co. v. Sacket (Kentucky), 190 Southwestern, 130, p. 141.

TITLE—ADVERSE POSSESSION.

Before a person can acquire an actual possession to coal land to which he had no title, or only as such grounds to claim ownership that merely gives him a color of title, he must enter upon the land with the intention of holding it. If he enters under a deed or other instrument, he will be in actual possession to the extent of his boundaries. The adverse possession necessary to create title must have for its basis the existence of physical facts, such as making improvements upon the land or doing such acts upon it as will openly evince a purpose to hold dominion over it in hostility to the title of the real owner, and such as will give notice to the real owner that the purpose is to hold it in hostility to his title, and such holding must continue for a period of 15 years under the statute of Kentucky.

Tennis Coal Co. v. Sacket (Kentucky), 190 Southwestern, 130, p. 135.

RELIEF FROM FRAUD—FALSE REPRESENTATION.

The statute of Washington provides that a cause of action for relief on the ground of fraud is not to be deemed to have accrued until the discovery, by the aggrieved parties, of the facts constituting the fraud (Remington Code, 1915, sec. 159). Under this statute a part owner of a mining property who was induced by the fraudulent representations of another part owner to join in the sale of the mining property at a stated price, where the purchaser was, in fact, offering a greater price, and the coowner conducting the negotiations falsely represented that the purchaser would pay no more than the stated price and that he would not deal with any other coowner, is entitled to sue and recover at any time within three years after the discovery of the fraud.

Langley v. Devlin (Washington), 163 Pacific, 395, p. 396.

SURFACE AND MINERALS—OWNERSHIP AND SEVERANCE.**SEPARATION OF SURFACE AND MINERALS—ADVERSE POSSESSION.**

The statute of Kentucky (section 2366a) provides that where the mineral or other interest passed from a claimant in possession of the surface of the land, the continuity of the possession of such mineral interest shall not be deemed thereby to be broken; but the possession of the surface by the original claimant from whom such mineral interest or rights passed shall be deemed to be the possession of the mineral interest for the benefit of the person, or his heirs, to whom the mineral interest shall have passed. Under this statute where the vendor of the coal was in possession of the land at the time of the sale of the coal interest, and therefore remained in possession until the adverse possession of the land was held for the statutory period, counting from the first acquiring actual possession of the land, then the title of his grantee of the coal was made valid. But, if before the statutory period of adverse possession had expired, the original vendor had abandoned possession of the land or had been evicted by the owner of the paramount title, then the title of his grantee, the present claimant, failed.

Tennis Coal Co. v. Sacket (Kentucky), 190 Southwestern, 130, p. 141.

POSSESSION OF MINERAL INTERESTS.

Section 2326a of the statute of Kentucky provides that the possession of the surface of land shall be deemed the possession of the mineral interests therein, for the protection of the owners of the minerals. But this statute makes no change in the character of the estate of the owners of mining leases, but only protects them, and the leasehold remains personalty notwithstanding the statute, and

does not make members of a partnership holding leases of mining interests joint tenants.

United Mining Co. v. Morton (Kentucky), 192 Southwestern, 79, p. 82.

RIGHT TO SUPPORT.

Where different strata of earth are owned by different persons and there is no contract or statute which affects their interest, the owner of the upper stratum has an absolute right to have his land supported in its natural position by the stratum below.

Audo v. Western Coal & Mining Co. (Kansas), 162 Pacific, 344, p. 346.

SUBSIDENCE OF SURFACE—LIABILITY.

A coal-mining company owned a tract of land underlain with coal. The company, after it had mined out and removed the coal from under a part of the land, sold the surface of the entire tract to a purchaser for a valuable consideration and conveyed the same by warranty deed, but reserved to itself the remaining coal and other minerals under the tract with the right to mine and remove the same. The purchaser knew that the coal had been mined from under a part of the premises conveyed but he had no knowledge of the manner in which the coal had been mined nor the adequacy of the supports or pillars to protect the surface. Under such circumstances the right to subjacent support for the surface was not waived by the purchaser but he is entitled to have his land supported in its natural condition, and the deed on its face conveyed the subjacent support for the reason that such support was a part of the real property therein described and there was nothing to indicate that support for the surface above the coal had been withdrawn but the purchaser had the right to assume that such support had been left, and the mere knowledge of the fact that the coal had been mined from a portion of the land does not constitute a waiver of the right to subjacent support conveyed by the deed. Under the deed the grantor, the coal company, was liable to the grantee for damages caused by the subsequent subsidence of the surface.

Audo v. Western Coal & Mining Co. (Kansas), 162 Pacific, 344, p. 346.

COAL AND COAL LANDS.

GENERAL CONTRACT TO SELL COAL—CONSTRUCTION AND COMPENSATION.

A corporation organized for the purpose of selling the products of a number of coal mines, but itself mining no coal, employed a third person "as its sole agent in the matter of the sale of its product within a stated territory." In the construction of this contract the

word "products" was not intended nor understood to mean the entire output of the mines represented. Looking at the contract as a whole the word "product" would appear to mean the coal which was turned over to the selling corporation for sale, or in respect of which the selling corporation in some manner represented the coal companies, but it can not be construed to include as well coal sold by the coal-mining companies themselves and upon which the selling corporation itself received no compensation.

Gatewood v. New River Consolidated Coal & Coke Co., 239 Fed., 65, p. 59.

AGREEMENT FOR HAULING COAL—EXCLUSIVE USE.

A firm, as the owner of coal lands and coal mines, agreed with a railroad company that it should put in a side track for them from its main track to their coal mine. It was agreed that the firm should furnish the right of way, do the grading and bridging from the main track to the mine, and pay a part of the first cost of ties. The railroad company was to furnish the material necessary for laying the tracks and to maintain them at its own expense. The consideration for the agreement was that the railroad company was to have exclusive use of the tracks and right of way for its use in handling the business of other industries except coal mines that might be located adjacent to the tracks or reached from this branch road and that it should conduct its business in a manner which would not interfere with the business of the coal firm. From the entire contract and the situation of the parties when the contract was made, it was the intention that the owners of the coal mine were to have a monopoly of the use of the tracks so far as coal-mine products were concerned in return for the large amount of money they expended in its construction, and the railroad company was to have exclusive use of the track or a monopoly of its use for all purposes except coal mines. Under the agreement and the acts of the parties and their construction thereof, and owing to the large amount of money invested by the mine owners in the right of way and railroad tracks, the railroad company has no right, without their consent, to transfer cars of coal from its main line and haul them over the branch line, to the interference of the business of the coal-mine owners.

Sholl Bros. v. Peoria, etc., Ry. Co. (Illinois), 114 Northeastern, 529, p. 532.

WIDOW'S DOWER IN MINES.

Where mines have been opened in the lifetime of the husband and the right of dower exists, the widow becomes seized of her right and is entitled to dower in mines opened and used during the lifetime of her husband. Where the dower interest can not be assigned by reason of the irregularities in the veins of coal and the premises are of

such nature that dower can not be assigned by metes and bounds, the court may order in lieu of such assignment that the widow be entitled to and receive one-third of the rents and profits of the mines opened and worked from the date of the decree.

Shupe v. Rainey (Pennsylvania), 100 Atlantic, 138, p. 141.

ASSIGNMENT BY HUSBAND FOR BENEFIT OF CREDITORS—WIFE'S INTEREST.

The owner of lands containing coal conveyed a certain part of the coal in place by deed in which his wife did not join. Subsequently the husband on his insolvency, by a deed or assignment for the benefit of his creditors, conveyed the surface of the land to an assignee by deed in which the wife joined. The joining of the wife in the deed or assignment in no way affected her interest in the coal previously conveyed by the husband and she still held her dower right in the coal.

Shupe v. Rainey (Pennsylvania), 100 Atlantic, 138, p. 139.

CONVEYANCE INCLUDING COAL—BREACH OF COVENANT—DAMAGES.

A deed conveying a tract of land and expressly including three acres of coal, with a general covenant of warranty includes in the covenant coal as well as the land; and the failure of the grantor to deliver the coal, it having been mined out and removed, is a breach of the covenant and entitles the grantee to sue for such breach and recover damages. In such case the measure of damages is the relative value of the coal mined out of the three acres before the plaintiff purchased, as compared with the value of the entire tract estimated with regard to the price fixed by the parties to the deed for the entire purchase; and it is proper for either party to give evidence of the peculiar advantages or disadvantages of the part lost with reference to the whole of the land.

Clark v. Steele (Pennsylvania), 99 Atlantic, 1001, p. 1003.

SALE OF COAL IN PLACE—REMOVAL—DAMAGES.

In an action for the breach of a covenant in a deed conveying certain described land and the coal in place under a certain part of the land, where it appeared that the coal had been mined out and removed without the knowledge of the grantee, the burden of proving the relative value of the part to which the title failed as compared with the value of the whole property was on the complainant. He had a right to show also, where it was demonstrated that the practical effect of the removal of the coal would be a subsidence of the surface and consequent injury to buildings, springs, etc., and under such circumstances the grantor is not limited in his damages to the market

value of the coal, but may include the probable damages to buildings, springs, and streams that have ensued or that are likely to ensue from the removal of the coal. Such proof is admissible under a general averment of the resulting damages as these would necessarily follow from the breach of the covenant and was in the contemplation of the parties where it was shown that both were familiar with the general effect on the surface of the removal of underlying coal, including the damage of the subsidence of the surface and consequent injury to buildings, springs, and streams. It will be presumed that the parties had these things in mind when they contracted for the sale of the land with the coal in place under the farm buildings on the premises.

Clark v. Steele (Pennsylvania), 99 Atlantic, 1001, p. 1003.

SALE OF MINERALS—SUBJACENT SUPPORT—WAIVER.

The right to subjacent support to the surface where land has been sold with the reservation of the underlying coal would not be deemed to have been waived or lost to the owner of the surface unless such clearly appears, from the language used in the conveyance to have been the intention of the parties, as an owner of the surface is entitled to actual absolute support unless that obligation is distinctly removed.

Ando v. Western Coal & Mining Co. (Kansas), 162 Pacific, 344, p. 346.

OIL AND OIL LANDS.

EFFECT OF PRESIDENT'S WITHDRAWAL ORDER.

In the absence of a discovery on an oil location on the public domain and in the absence on the part of the locator of diligent prosecution of work leading to discovery, even though in actual possession of the claim, as against the Government, he is subject at any time to the possibility of a withdrawal of the privileges offered to him and consequently a termination of his rights.

United States v. McCutchen, 238 Fed., 575, p. 579.

OFFER AND ACCEPTANCE.

The Government by its mining statutes offers to qualified persons the minerals in the public domain through the means of mining locations, and if the offer so made is accepted by compliance and by the location of a valid mining claim in accordance with the statutory provisions before the offer is withdrawn by the Government, it can not, after acceptance, withdraw its order, as the offer then becomes binding and an enforceable obligation. But the Government may withdraw its offer at any time before acceptance and the situation stands as if no offer had ever been made.

United States v. McCutchen, 238 Fed., 575, p. 583.

LANDS VALUABLE FOR OIL.

The act of Congress February 11, 1897 (29 United States Statutes at Large, p. 526), provides that any person authorized to enter land under the mining laws may enter and patent lands containing petroleum or other mineral oils, and chiefly valuable therefor, under the provisions of the statute relating to placer mineral claims. Congress intentionally limited the right of entry upon and location of oil lands to such lands as were "chiefly valuable therefor," meaning thereby land chiefly valuable for oil. The Government as sovereign proprietor had a right to say what lands might be entered and located for oil. It could deny the right to enter upon any such land and could limit the land upon which entry might be lawfully made; and as to any land not within the category specified in the statute there was no invitation or authorization given to enter them and no right could be obtained as against the Government by so doing.

United States v. McCutchen, 238 Fed., 575, p. 583.

WITHDRAWAL ORDER.

The President's withdrawal order of September 27, 1909, was effective to withdraw from location, entry, exploration, and exploitation land attempted to be located as an oil location, but upon which the locator had made no discovery of oil and was not in a diligent prosecution of work for the purpose of discovery, prior to or at the time of the withdrawal order.

United States v. McCutchen, 238 Fed., 575, p. 593.

CONTRACT OF SALE—CONSTRUCTION—RIGHT OF POSSESSION.

A contract for the sale of certain oil lands provided that the purchaser should not remove nor cause to be removed any of the pipe or casing in any well drilled on the property; and it also provided that in the event of a forfeiture by the purchaser of the right to purchase the land under the contract, the casings and derricks should revert to the vendor. The contract did not in express terms give the purchaser the right of possession of the property, but in view of the wording of the contract and of the subsequent conduct of the parties to the contract, immediate entry by the purchasing company upon the land and the development of oil thereon were within the contemplation of the parties. Such contracts may by implication give the right of immediate possession by the vendee where no express provision to that effect is contained in the contract of sale.

Francis v. West Virginia Oil Co. (California), 162 Pacific, 394, p. 395.

EMINENT DOMAIN.

RIGHT OF WAY OVER MINING CLAIM—DAMAGES.

A railroad company appropriated a strip of land for a right of way over a mining claim. The mining company was entitled to recover whatever damages it might suffer by reason of the appropriation of the right of way and the railroad company could not escape liability or mitigate the damages by permitting or offering to permit the mining company to use a part of the appropriated land for dumping purposes.

Bingham, etc., R. Co. v. North Utah Mining Co. (Utah), 162 Pacific, 65, p. 68.

EASEMENTS—USE OF SURFACE.

Whether an easement in connection with mineral land and mining rights is appurtenant or in gross is to be determined mainly by the nature of the right and the intention of the parties creating it. If an easement granted is in its nature an appropriate and useful adjunct of the land conveyed, having in view the intention of the grantee as to its use, and there is nothing to show that the parties intended it to be a mere personal right, it will be held to be an easement appurtenant to the land and not in gross. Under this rule where a deed conveyed the minerals in a tract of land with the privilege of using the surface for rights of way for tramways or other means of transportation necessary for the removal of the minerals as well as the minerals from other lands, it will be considered that the minerals so conveyed should be mined or produced in connection with the minerals from other lands, owned or acquired by the grantee and the easement created by the grant of the rights of way will be appurtenant to the grant of the minerals contained in the land, it being reasonably necessary that the minerals in such land should be mined in connection with the minerals from other lands. The grant of such an easement will confer upon the grantee the right to construct a tramroad across the surface of the land containing the minerals granted, for the purpose of hauling timber across the same to be manufactured into lumber to be used in mining the minerals granted, as well as the minerals produced by the grantee from other lands in connection with the minerals so granted.

Jones v. Island Creek Coal Co. (West Virginia), 91 Southeastern, 391, p. 393.

EASEMENT—RIGHT OF WAY FOR HAULING.

The purchaser of coal in place from the owner of the surface, who in connection with the purchase of the coal obtained an ease-

ment over the surface for mining operations both as to the coal purchased and as to coal owned and mined by him in other lands, together with the right to haul or transport lumber manufactured from timber on his lands and used in his mining operations, will not be prohibited from hauling all of such lumber across the land over which he has the easement upon a tramroad constructed by him and he may haul and sell the excess of lumber above that actually needed for mining purposes where this is merely incidental to the main business of producing minerals from his other lands.

Jones v. Island Creek Coal Co. (West Virginia). 91 Southeastern, 391, p. 394.

MINING TERMS.

JUDICIAL NOTICE—INSTANCES.

The courts of Kansas take judicial notice of the following mining terms: Air courses are passages for conducting air. Entries are those places in coal mines used by the miners and other workmen generally in going to and from their work and through which coal is hauled from the necks of the rooms to the foot of the shaft. Room is a place in which a miner works and from which he mines coal. Traveling ways are places for the passage of workmen to and from different parts of the mine.

Ricardo v. Central Coal & Coke Co. (Kansas), 163 Pacific, 641, p. 643.

GIN MEN.

Gin men are persons employed in a mine to clean up and remove fallen slate or rock or other material from the roof.

North Jellico Coal Co. v. Stewart (Kentucky), 191 Southwestern, 451, p. 453.

JACK POLE.

A jack pole is an iron rod or pole sharpened at each end and one end is inserted in the roof and the other in the floor of the mine in notches made for this purpose. A wire cable runs from the jack pole to the coal cutting machine and the pole operates as a stay to hold the machine in place during its operation, and by reason of the weight and the force necessary to run the coal cutting machine the jack pole and the wire cable are subjected to a strain of something like 4,000 pounds.

Tennessee Coal, Iron & Railway Co. v. Wiggins (Alabama), 73 Southern, 516, p. 517.

MACHINE HELPER.

A machine helper is a man who assists a machine runner and his duty is to set up and hold the iron rod or jack pole which runs a wire cable to the machine, and he is subject to the orders of the machine runner.

Tennessee Coal, Iron & Railway Co. v. Wiggins (Alabama), 73 Southern, 516, p. 517.

MACHINE RUNNER.

A machine runner is a man who operates the levers which control the electric power driving a coal cutting machine and he determines the movements of the machine while in operation.

Tennessee Coal, Iron & Railway Co. v. Wiggins (Alabama), 73 Southern, 516, p. 517.

MINE.

Under the Alabama code the term "mine" applies alike to ore and coal mines.

Sloss-Sheffield Steel & Iron Co. v. Bearden (Alabama), 74 Southern, 230, p. 231.

SOLLERS.

Sollers are intermediate platforms between the levels in a ladderway, the purpose of which is to stop anything falling through the ladderway, or to catch a man's body in case he falls.

Drajcevic v. Champion Copper (Michigan), 160 Northwestern, 451.

MINING CORPORATIONS.

BANKRUPTCY—SALE OF PROPERTY—RIGHTS OF STOCKHOLDERS.

Where the property of a bankrupt mining corporation was sold on foreclosure proceedings, the referee may properly refuse to confirm the sale, where between the time of the sale and confirmation it was shown that the property had greatly appreciated in value and was then worth more than sufficient to pay all the debts of the corporation. Under such circumstances it is proper to give the stockholders time to form a committee to purchase the property under an order providing that no stockholders' committee should be allowed to bid except upon the condition that all the indebtedness and administrative expenses should be paid, but making the sale competitive and authorizing the property to be sold to the committee or person bidding the highest figure.

Ohio Copper Mining Co., In re, 237 Fed., 490, p. 493.

POWER TO ENCUMBER MINING PROPERTY.

The statute of Colorado (R. S. 1908, sec. 865) prohibits manufacturing corporations from encumbering their mining property, or the principal machinery incident to the production, unless approved by a majority of the stockholders. Under this statute a lease is held to be an encumbrance and a lease of mining property made by a mining corporation without a legal vote of the majority of the stockholders approving it is voidable by the stockholders, and the right of action to avoid it is in them and not in the corporation.

Elder v. Western Mining Co., 237 Fed., 966, p. 971.

POWER TO ENCUMBER PROPERTY—WANT OF VALIDATION BY STOCKHOLDERS.

A statute prohibiting a mining corporation from encumbering its property without the consent of a majority of the stockholders can not be construed to make valid a lease made by the directors until it has been repudiated by a majority of the stockholders; but until a lease has been submitted at a proper and legal meeting of the stockholders and a majority of all the shares of stock have voted in favor of the lease, the lease is voidable by any stockholder and any such stockholder may maintain a suit to set it aside without the assent of the majority of the stockholders or of any stockholder other than himself to a repudiation of the lease.

Elder v. Western Mining Co., 237 Fed., 966, p. 973.

AUTHORITY OF PRESIDENT.

The president of an oil company has no power ordinarily to render the company liable by the execution of a note in its name payable to himself; but where the by-laws of the corporation give its president and treasurer authority to sign such notes as he might find necessary for the procurement of funds for the corporation, and resolutions of the directors were passed from time to time authorizing the president and treasurer to borrow for the use of the corporation such funds as he thought necessary, this constitutes such conduct on the part of the corporation and its officers as to give authority to the president to borrow from himself as well as others and to execute notes of the corporation to himself for the loans made. The knowledge of the corporation of the acts of the president in borrowing money from himself and executing the notes of the corporation therefor and the use of the money so borrowed on the authority of the directors constitute a complete ratification of the acts of the president and renders the corporation liable on such a note.

Eastman Oil Co., In re, 238 Fed., 416, p. 418.

INSOLVENCY—DISTRIBUTION—INTEREST.

Where a mining corporation is insolvent and all the claims are of equal dignity, the dividend and distribution are made upon the basis of the amount of principal due at the time the property passed into the hands of the court; but where there are claims of different classes or rank and no class or rank is secured by mortgage of real estate the mortgagee is entitled to the principal and to the interest that accrues up to the time of the satisfaction, although in the distribution nonlien creditors may receive no dividend whatever.

Spring Coal Co. v. Keech, 239 Fed., 48, p. 53.

AUTHORITY OF DIRECTOR.

The director of a mining corporation has no authority by virtue of his office to make a mortgage disposing of all of the corporation's property and disabling it from carrying on its business in the absence of any previous exercise of such authority or custom or the holding out of him by the corporation as having such authority.

Danglade & Robinson Mining Co. v. Mexico-Joplin Land Co. (Missouri App.), 190 Southwestern, 35, p. 37.

AUTHORITY OF GENERAL MANAGER.

A director of a mining corporation who was elected "to manage the mining interests and property of the company," but whose business as such manager was confined to the physical management of the property and the carrying on of the mining operations, and who as

treasurer had the right to collect in money belonging to the corporation and to disburse the same, including the payment of debts, has no implied authority to mortgage or pledge the machinery of the corporation to pay or secure a debt of the corporation.

Danglade & Robinson Mining Co. v. Mexico-Joplin Land Co. (Missouri App.), 190 Southwestern, 35, p. 37.

MORTGAGE TO SECURE DEBT—CONVERSION—SET-OFF.

A director and general manager of a mining corporation without authority executed a mortgage on certain mining machinery to secure the debt of the company due and owing as royalty on the ore mined. In an action by the mining company against the mortgagee and creditor for conversion of the mining property, it was held by the court that the mining company's debt was so closely connected with the creditor and mortgagee's conversion of the property that the one should be set-off against the other, and that the mining company should recover the difference only between the debt and the reasonable value of the property converted.

Danglade & Robinson Mining Co. v. Mexico-Joplin Land Co. (Missouri App.), 190 Southwestern, 35, p. 38.

INSURABLE INTERESTS IN MINING PROPERTY—JUDGMENT LIENS.

A mining corporation purchased certain mining property subject to the lien of a judgment taken against the seller. The mining company as such purchaser had an insurable interest in the property and caused the same to be insured for its benefit. On the destruction of the property by fire, on the collection of the insurance money by the purchasing corporation, the fund thus derived from the insurance was not subject to the lien of the judgment and did not stand in its stead, and when the property was destroyed the security for the judgment was wiped out.

Vogelstein v. Athletic Mining Co. (Missouri App.), 192 Southwestern, 760, p. 763.

LIABILITY ON PURCHASE OF MINING PROPERTY.

A mining corporation by merely purchasing material and machinery of another company does not assume any liability under a judgment rendered against the selling company and to which the purchasing company was not a party except that it takes the fixtures and property subject to the judgment lien. The judgment creditor under such circumstances can proceed only against the property.

Vogelstein v. Athletic Mining Co. (Missouri App.), 192 Southwestern, 760, p. 763.

STATUTE MAKING REMOVAL A CRIME—VALIDITY.

The statute of Arizona (Civil Code Arizona, 1913, par. 2228) attempting to prevent the removal of a suit from a State court to a

Federal court by a foreign corporation by making an attempt to so remove a case a misdemeanor is unconstitutional and void.

Copper Queen Consolidated Mining Co. v. Jones, 237 Fed., 131, p. 134.

See *Deutsch v. Alaska-Gastineau Mining Co.*, 237 Fed., 215.

See *Key v. West Kentucky Coal Co.*, 237 Fed., 258.

MATTERS DETERMINED ON PETITION FOR REMOVAL.

In a proceeding on the part of a mining corporation to remove a cause from a State court or to remand to a State court an action removed to a Federal court, the question whether the facts are properly pleaded or sufficiently pleaded to constitute a cause of action is not for the court to determine upon the hearing.

Deutsch v. Alaska-Gastineau Mining Co., 237 Fed., 215, p. 218.

See *Copper Queen Consolidated Mining Co. v. Jones*, 237 Fed., 131, p. 135.

Key v. West Kentucky Coal Co., 237 Fed., 258.

PETITION AND BOND OPERATE AS REMOVAL—WAIVER.

When a petition for the removal of a case is sufficient, on the filing of proper bond by a mining corporation in an action in a State court, no order of the court removing the case to a Federal court is necessary, as the filing of the petition and the giving of the bond, ipso facto, removes the case. A defendant mining company loses none of its rights by following the case in the State court after the proper petition and bond have been filed and removal denied.

Nelson v. Black Diamond Mining Co., 237 Fed., 264, p. 266.

REMOVAL—FILING SECOND PETITION.

The removal statutes do not authorize or permit the filing of a second petition for removal except where, before a trial on the merits, the plaintiff changes the parties by a dismissal of the local defendant, or where before or even during the trial the plaintiff amends his pleading and thereby increases the amount in controversy to a sum exceeding \$3,000, exclusive of interest and costs, or where the plaintiff does some other things analogous to these mentioned.

Key v. West Kentucky Coal Co., 237 Fed., 258, p. 263.

REMOVAL—SEPARABLE CONTROVERSIES.

In an action by a miner against a mining corporation and its managing agent for injuries received by the alleged negligence of such managing agent, there is no separable controversy as the parties defending are properly joined and there can be no removal of the case from a State to a Federal court on that ground.

Key v. West Kentucky Coal Co., 237 Fed., 258, p. 261.

See *Deutsch v. Alaska-Gastineau Mining Co.*, 237 Fed., 215.

Nelson v. Black Diamond Mining Co., 237 Fed., 264.

LIABILITY FOR ATTORNEY FEES.

A mining corporation is liable for the fees and services of an attorney employed by the promoters of the corporation before the certificate of incorporation was filed, but where the employment was made after the articles of incorporation had been executed and the directors named therein were present and for several years thereafter the same members continued to fill the positions as directors for the company and where the services were rendered after the corporation was organized and the certificate duly filed and recorded.

Expansion Gold Mining & Land Co. v. Campbell (Colorado), 163 Pacific, 968, p. 970.

MINING PARTNERSHIPS.

NATURE.

A mining partnership to which the parties do not by contract give the ordinary incidents of commercial partnerships is distinguishable from the ordinary commercial or trading partnership in characteristics which flow from the fact that in mining partnerships there is no *delectus personae* except as to the few peculiarities which depend upon this distinction. The law governing a mining partnership is not different from that applicable to a commercial partnership, and the elements of the latter are common also to the former.

United Mining Co. v. Morton (Kentucky), 192 Southwestern, 79, p. 83.

EXPLORING OIL TERRITORY.

A partnership formed for the purpose of engaging in exploring prospective oil territory is not a mining partnership as defined by the statute of California (Civil Code, sec. 2511), where the members are associated together for the purpose of working a mining claim.

Callahan v. Danziger (California App.), 163 Pacific, 65, p. 67.

CONTRACT FOR OPERATING MINE.

A contract under which the owners of mineral lands and a mining company agreed to operate them is in the nature of a partnership agreement where, instead of the usual royalties, the lease prescribed the mode in which the mine is to be worked by controlling the salaries of the officers and the operating expenses, and providing that the mine should be operated in a certain stated manner and that the owners of the land were to receive at stated intervals each year "as in the nature of royalties 40 per cent of the net profits." This manner of paying for the privilege of mining is not essentially different from paying a certain amount for each ton taken out, and the amounts received under the agreement should be held to be rents and profits from the land.

Pettit's Estate, In re (Minnesota), 161 Northwestern, 158, p. 161.

CONSTRUCTION OF CONTRACT—PRACTICAL CONSTRUCTION.

The parties to a contract are supposed to know best what they mean, and if in reducing it to writing, words or terms were used that rendered the contract ambiguous or uncertain, then the construction by the parties as shown by their acceptance and acts thereunder can be of value in aiding the court in ascertaining the true intention and

meaning of the instrument, and the construction placed upon the contract by the parties themselves should be upheld by the courts.

Sholl Bros. v. Peoria, etc., Ry. Co. (Illinois), 114 *Northeastern*, 529, p. 531.

ORAL CONTRACT—STATUTE OF FRAUDS.

Two persons orally formed an equal partnership for the purpose of taking mining options and leases and acquiring and developing mining rights and holdings in certain counties in Kentucky. A partnership agreement to deal in mining leases is an agreement relating to personal property and not to real estate, as leaseholds are personalty, and such an agreement resting in parol is not within the statute of frauds.

United Mining Co. v. Morton (Kentucky), 192 *Southwestern*, 79, p. 82.

DEALING IN REAL ESTATE.

There is nothing in the nature of mining which forbids the creation of a partnership by an ordinary partnership contract which would draw to the relation of the parties the incidents of a trading partnership and destroy the distinctions which are based upon the non-existence of the *delectus personae* in strict mining partnerships. A mining partnership agreement made by parol, although it necessarily required the acquisition of an interest in land, may be implied from the acts of those sought to be charged as partners.

United Mining Co. v. Morton (Kentucky), 192 *Southwestern*, 79, p. 83.

ABANDONMENT.

One partner in a mining partnership agreed to devote his entire time to the partnership business. Subsequently he left the State, but returned once at the expense of the other partner, and thereafter for a period of at least five years paid no attention whatever to the partnership business and did not even inform his partner of his whereabouts. This was a complete abandonment of the partnership business. On such abandonment the other partner is necessarily left to the management of the partnership business and may surrender mining leases, or he may make a general assignment for the benefit of the firm's creditors.

United Mining Co. v. Morton (Kentucky), 192 *Southwestern*, 79, p. 83.

PARTNERSHIP AGREEMENTS.

A partnership formed for the purpose of dealing in mining leases does not constitute the members cotenants in the leases acquired, and one partner in such a partnership may sell and assign a mining lease acquired by the firm and thereby bind his partner and the partnership.

United Mining Co. v. Morton (Kentucky), 192 *Southwestern*, 79, p. 83.

CREATION OF PARTNERSHIP RELATIONS.

A partnership relation between parties engaged in acquiring mining properties for their joint benefit must exist at the time such properties are acquired by one of the parties to such an arrangement in order to entitle the other to an interest therein.

Hollingsworth v. Tufts (Colorado), 162 Pacific, 155, p. 159.

INTEREST IN PROPERTY.

A person claiming an interest in mining property as a partner under an agreement that mining property discovered and located should be held and owned by the partners must show that the mining property in which he is seeking to establish an interest was discovered and located under and pursuant to the partnership agreement.

Hollingsworth v. Tufts (Colorado), 162 Pacific, 155, p. 159.

MONOPOLY AGREEMENT WITH RAILROAD.

An agreement between a partnership owning coal land and coal mines and a railroad company whereby the owners of the coal land and coal mines, by reason of their contributions and aid in the construction of a railroad over their own lands, was to have a monopoly of the road as to the matter of hauling and transporting coal, is not contrary to public policy and void, as the side track or branch road contemplated is not a part of the public railroad system of the company and could not become a part thereof, unless it bought, purchased, or acquired the same by condemnation proceedings.

Sholl Bros. v. Peoria, etc., Ry. Co. (Illinois), 114 Northeastern, 529, p. 532.

MINING CLAIMS.

NATURE AND GENERAL FEATURES.

RIGHTS AND INTERESTS.

A prospector who was under a grubstake contract to locate mining claims, in company with another prospector who was under a grubstake contract from another party to locate mining claims, located different claims for their respective principals. A letter written by one of the prospectors to his principal to the effect that "we discovered and located" certain mentioned claims is not sufficient proof to show that the principal of the prospector who wrote the letter had an interest in a particular claim located by the second prospector in the name of his principal. The statement that "we discovered and located" is not inconsistent with the theory that he merely assisted the other grubstake prospector in locating the claim. The fact that the writer did assist the other prospector and in fact wrote the location notice and witnessed and recorded the same will not, of itself, subject the location to the grubstake contract entered into with him.

Turner v. Wells, 238 Fed., 766, p. 769.

NATURE AS REAL PROPERTY—ORAL AGREEMENTS.

The statute of Oregon (sec. 5132 L. O. L.) provides that a mining claim shall be deemed real property, and it follows as a corollary that an enforceable interest in a mining claim can not be created by a verbal option.

Grand Prize Hydraulic Mines v. Boswell (Oregon), 162 Pacific, 1063, p. 1067.

ASSISTING IN LOCATION—ESTOPPEL.

A miner and prospector who went upon and prospected and made some development of a tract of land, and who subsequently assisted another miner and prospector in surveying and laying off and staking another claim, is estopped from asserting title or right to such later claim after the discovery of valuable minerals thereon, on the ground that the discovery point and the part of the claim surrounding such point was located within the limits of the ground claimed by him. The same rule of estoppel applies to the son of the first locator, who, in the matter of making the second location, was present and acquiesced in all that his father did and who knew that the work of such second locator was done within the boundaries of the original claim taken or staked by his father.

Grand Prize Hydraulic Mines v. Boswell (Oregon), 162 Pacific, 1063, p. 1066.

VEIN OR LODE.**VEIN—WHAT CONSTITUTES.**

A vein or lode comes within the meaning of the United States mining statutes so long as there is a fissure or gouge or any evidence of mineralization which will lead a practical miner from one ore body to another and which does in the course of his work so lead him.

Alameda Mining Co. v. Success Mining Co. (Idaho), 161 Pacific, 862, p. 865.

LODE—WHAT CONSTITUTES.

Tilted beds of sedimentary strata containing ore would by the geologist be called beds and not lodes; but the intent of the United States mining statutes is not to make distinctions based upon genetic principles. What the geologist might call beds of ore the courts may find to be lodes within the meaning of the United States mining statutes authorizing the location of mining claims upon the public domain.

Alameda Mining Co. v. Success Mining Co. (Idaho), 161 Pacific, 862, p. 865.

APEX OF VEIN.**TERMINAL EDGE OF VEIN.**

An apex is the top or terminal edge of a vein on the surface or the nearest point to the surface, and must be the top of the vein proper, rather than of a spur, and must be a point from which the vein has a dip as well as a strike.

Alameda Mining Co. v. Success Mining Co. (Idaho), 161 Pacific, 862, p. 865.

DISCOVERY.**NO RIGHTS ACQUIRED WITHOUT DISCOVERY.**

A person who enters upon the public domain and locates land for its mineral contents, as oil lands, though he may erect appropriate monuments and post and properly file location notices, if he makes no discovery of minerals or oil, he acquires no right of any nature against the Government or any private individual, save the right to proceed with diligence to effect an actual discovery of minerals, gas, or oil.

United States v. McOutchen, 238 Fed., 575, p. 579.

STATUS IN ABSENCE OF DISCOVERY.

The status of a locator of a mining claim in the absence of a discovery is in the nature of a tenant at sufferance.

United States v. McOutchen, 238 Fed., 575, p. 580.

MARKING BOUNDARIES.**COMPLIANCE WITH STATUTE.**

As against a subsequent locator who has actual knowledge of the existence of a mining claim and who had assisted in performing the assessment work for the year previous to his attempted relocation, it is sufficient that the original location was distinctly marked on the ground so that its boundaries could be readily traced.

Gold Creek Antimony Mines & Smelter Co. v. Perry (Washington), 162 Pacific, 996, p. 999.

MARKING BOUNDARIES UPON THE GROUND.

The United States mining statutes (R. S. 2324) require a location to be distinctly marked upon the ground, with the name of the locator, the date of location and such references to natural objects or permanent monuments as will identify the claim. These provisions are supplemented by the statute of Washington (Remington Code, sec. 7379). The purpose of the United States statute and of the State statute is to give notice to prospectors who are looking for mineral locations of what has been already appropriated in order that they may govern themselves accordingly. It is also the purpose to prevent fraud by swinging or floating. In accomplishing these purposes, courts are inclined to be liberal with persons making mining locations, and are not inclined to defeat a claim of a locator who has in good faith attempted to comply with the requirements of the law by technical criticism of the act relied upon to constitute a valid location.

Gold Creek Antimony Mines & Smelter Co. v. Perry (Washington), 162 Pacific, 996, p. 997.

DESCRIPTION.**SURVEY—MONUMENTS CONTROL DISTANCE.**

Where mining claims were actually surveyed out on the ground and the four corners established and if in the application of descriptions in the patent to the claims as surveyed, a latent ambiguity arises in that a distance called for conflicted with the corners as established for the mining claims, then the mining claim's corners as fixed control the call for distance as given in the patent and the distance called must yield to the established corners of the claims.

Plummer v. McLain (Texas Civil App.), 192 Southwestern, 571, p. 575.

ASSESSMENT WORK.**TIME OF PERFORMANCE—RESUMING WORK.**

The United States mining statutes (sec. 2324) require that all locators of mining claims perform \$100 worth of work during each year to entitle them to hold the claim as against a relocater. After the year in which a location is made the entire assessment work must be performed during each year and must be completed within each

calendar year, or a third person may enter and relocate the claim, unless the locator has resumed work after his failure to complete the same before a relocation is made. If a locator or owner has begun the assessment work before the expiration of any given year and is carrying on to completion such work the claim is not subject to relocation, although the locator or owner is not on a particular day upon the claim at work.

Plough v. Nelson (Utah), 161 Pacific, 1134.

PROOF OF FAILURE TO PERFORM ASSESSMENT WORK.

A person who has been acquainted with a mining location for years, familiar with its workings, and who assisted in doing the assessment work for the previous year, must make a strong case of failure of the prior locator to perform the annual assessment work before he can be heard to say the ground was vacant and unoccupied and subject to relocation by him.

Gold Creek Antimony Mines & Smelter Co. v. Perry (Washington), 162 Pacific, 996, p. 997.

EXTRALATERAL RIGHTS.

TERMINATION OF VEIN.

The fact that a vein terminates against a granite or monzonite at one end does not affect the extralateral rights given by the provisions of section 2322 of the United States Revised Statutes. Where a vein so terminates the locator would be entitled to have the end line pass through such point of termination parallel with the vertical plane of the other end line, thus giving him the extralateral right of the pursuit of the vein between the planes bounded by these end lines beneath all other mining claims under which it dips.

Alameda Mining Co. v. Success Mining Co. (Idaho), 161 Pacific, 862, p. 865.

BASES—RIGHT TO FOLLOW VEIN.

The provisions of section 2322, United States Revised Statutes, is determined by the apex on the surface upon which the prospector makes his location and the dip of the vein, and not upon the levels in the depth of the earth opened and disclosed in the working of the mine.

Alameda Mining Co. v. Success Mining Co. (Idaho), 161 Pacific, 862, p. 866.

COURSE OF VEIN.

The course of a vein is not determined by its direction at any single given point where the vein is a crooked one. A locator's extralateral rights must be determined by the course of the vein at its apex at the surface of the claim. The most practical rule is to

regard the course of the vein as that which is indicated by the surface outcropping or surface exploration and workings. The lower levels of a mine frequently show a different direction of the vein from that which guided the miner in making his location and are at variance with conditions shown in openings nearest to the surface.

Alameda Mining Co. v. Success Mining Co. (Idaho), 161 Pacific, 862, p. 866.

EXPERT AND POSITIVE TESTIMONY.

The positive testimony of miners who mined the ore and developed the mine and the engineers and others who made actual surveys of the mine involved in a controversy as to extralateral rights must be taken for more than the speculative theories of experts on the geology and formation of ore bodies and the mineralization of veins. Physical facts should be given greater weight than mere expert opinion and speculative theories.

Alameda Mining Co. v. Success Mining Co. (Idaho), 161 Pacific, 862, p. 868.

RELOCATION.

KNOWLEDGE OF PRIOR LOCATION—RIGHTS OF RELOCATOR.

A relocater can not base his right to make a valid relocation of an existing mining claim on the ground of the insufficiency of the original location notice, where he was not deceived or misled by any false or deficient description and where it plainly appears that he knew the boundaries of the original claim and entered within them for the purpose of acquiring for himself the benefit of the prior locator's labor and expenditures, believing that such prior locator had forfeited his rights, not in ignorance of such rights or for want of sufficient description of the property in the location notice. The purpose of a description in a location notice is to give notice of the location and where a relocater has actual notice he is not in a position to complain of technical defects which in no way affect his rights.

Gold Creek Antimony Mines & Smelter Co. v. Perry (Washington), 162 Pacific, 996, p. 997.

RELOCATION AS ABANDONED PROPERTY—NOTICE.

The statute of Washington (Remington Code, sec. 7365) requires the notice of relocation of a mining claim to state if the whole or any part of the new location is located as abandoned property. A relocater who makes his relocation upon ground known to be embraced in a former location, basing his right upon the fact that his location is made upon vacant and unoccupied ground and because of the failure of the prior locator to perform the annual assessment work, can not uphold the validity of such relocation because of the failure to comply with the State statute.

Gold Creek Antimony Mines & Smelter Co. v. Perry (Washington), 162 Pacific, 996, p. 998.

VACANT AND UNOCCUPIED GROUND—NOTICE.

A person with knowledge of the boundaries of an existing mining claim and with knowledge that the annual assessment work for the previous year had not been performed can not make a relocation on the theory that the ground is vacant and unoccupied, in the face of the statute of Washington (Remington Code, sec. 7365), which requires a relocation notice to state that the whole or any part of such new location is located as abandoned property.

Gold Creek Antimony Mines & Smelter Co. v. Perry (Washington), 162 Pacific, 996, p. 998.

POSSESSORY RIGHTS.**LOCATION AND DISCOVERY ESSENTIAL.**

Possession and enjoyment of mining claims under the United States mining statutes depend upon location and discovery of valuable minerals therein.

United States v. McCutchen, 238 Fed., 575, p. 579.

RIGHTS OF LOCATOR WITHOUT DISCOVERY.

A person after the location of a mining claim on the surface of the public domain may remain out of possession of the land and may await developments either by himself or of others on adjoining or in regional parcels, with no risk other than that of being dispossessed by the Government or some other locator. If this is not done he may return at some considerable period thereafter, or at any time prior to a location by another or to an actual withdrawal by the Government, and proceed to prosecute with diligence his search for minerals; and if he does so return and begins the work of discovery during the time he may be in possession actually engaged in the diligent prosecution of work leading to a discovery, he will be protected against inroads upon his right asserted either by the Government or by private parties, and if he ever actually effects a discovery of mineral, his vested right to the possession and enjoyment of the property and of its mineral contents may, in so far as may be necessary to secure protection to his rights, relate back to the time of his original location and will continue in the future for such time as he may comply with all valid laws and mining regulations. He is under such circumstances in the position of one who having made a discovery of minerals upon unappropriated public land has perfected the location thereof in the strictest sense of the term, and is thereafter subject to all the obligations and possessed of all the privileges of one in possession of a valid and subsisting mining claim.

United States v. McCutchen, 238 Fed., 575, p. 579.

ACTION TO RECOVER—STATUTE OF LIMITATIONS.

An action to recover a mining claim sold at an invalid tax sale must, under the present statutes of Nevada, be brought within two years from the time the right of action accrues, unless the person entitled to sue is under a statutory disability, and in such case the person has two years within which to bring the suit after the removal of the disability. This rule applies to patented and unpatented mining claims alike. The rule as stated is that "actions for the recovery of mining claims, or for the recovery of the possession thereof, must be commenced within two years from the time at which the plaintiff or those through or from whom he claims were seized or possessed of such mining claim, whether the same be patented or unpatented."

Wren v. Dickson (Nevada), 161 Pacific, 722, p. 736, December, 1916.

ASSIGNMENT—RIGHT OF ASSIGNEE.

The assignee of a mining claim can assert no right or title superior to that of his assignor, unless he was an innocent purchaser for value.

Grand Prize Hydraulic Mines v. Boswell (Oregon), 162 Pacific, 1063, p. 1067.

LOCATION ON STATE LANDS.

The right to locate a mining claim on lands owned by the State of Oregon is granted in order to facilitate the advantageous sale of the land. The locator of a mining claim on State lands acquires only a temporary possessory right, subject to be divested by a failure to purchase or lease the property in accordance with such reasonable regulations as the State land board may prescribe.

Grand Prize Hydraulic Mines v. Boswell (Oregon), 162 Pacific, 1063, p. 1064.

ADVERSE CLAIM.**ABANDONMENT OF APPLICATION—EFFECT OF JUDGMENT.**

After the filing of an adverse claim the applicant for patent withdrew his application and offered no evidence in opposition to the adverse claim. Upon instructions by the court the jury returned a verdict for the plaintiff and a judgment was entered accordingly. The judgment was simply for possession and not to form a basis for patent for the ground in conflict. By the withdrawal upon the part of the applicant of his application for patent the Government could have no further interest in the result of the suit, as title to the ground in conflict was not to be finally determined as a result of the verdict and judgment, and the action therefore became one merely for the right of possession for that part of the premises in conflict. But if the proof fails to disclose a conflict between the locations, the adverse

claimant can only claim such territory as is included within the boundaries defined by his location certificate and as there is no conflict he is not entitled to a judgment.

Lucky Four Gold Mining Co. v. Bacon (Colorado), 163 Pacific, 862, p. 863.

PATENTS.

COLLATERAL ATTACK.

A patent for a mining claim is not subject to collateral attack on the ground that the application for patent showed that the tract of land, or the claim patented, did not contain any of the valuable minerals.

Plummer v. McLain (Texas Civil App.), 192 Southwestern, 571, p. 577.

TUNNEL LOCATION.

DUMPING MATERIAL ON MINING CLAIM.

A tunnel claimant in the construction of his tunnel has originally no right to dump the waste rock and débris taken from his tunnel in the construction thereof upon the surface of another mining claim. In order to sustain the right as an implication that the law gives to him, he must show that the situation is such that he could not obtain without unreasonable labor and expense any other place or way to dump the material, and that he could not at an outlay of an amount that is within reason obtain dumping privileges elsewhere in the operation of the tunnel.

Himrod v. Fort Pitt Mining & Milling Co., 238 Fed., 746, p. 748.

ADVERSE POSSESSION.

The original locators of a mining claim constructed and used a tunnel in connection with the operation of the claim. The locator ceased using the tunnel for any purpose and abandoned the same in 1880. In 1887 another person took possession of the tunnel and constructed a flume or conduit therein to conduct the water therefrom for domestic purposes and erected obstructions or bars at the mouth or portal of the tunnel by which all other persons were entirely excluded from the use of the tunnel, and for more than 29 years the tunnel has been so used to the exclusion of all other persons. Such open, notorious, and exclusive possession and use of the tunnel for such a length of time was sufficient to give the person in possession title to the tunnel as against a grantee or assignee of the original locator and constructor.

Wall v. United States Mining Co., 239 Fed., 90, p. 92.

OIL LOCATION.

DISCOVERY ON OIL LOCATIONS.

With respect to oil lands arising out of the necessities of the case, discovery may and often must follow location, but upon discovery when ever attained in the absence of intervening rights of a superior nature, the same rights and results flow as if the discovery had preceded location, and pending discovery, the locator, after location, possesses all of the substantial rights consequent upon a discovery itself, as long as he continuously engages himself diligently in searching for oil upon the claim.

United States v. McCutchen, 238 Fed., 575, p. 579.

DELAY IN DISCOVERY—RELATION.

A valid location of an oil claim was made in 1900, and was not void because in fraud of the Government's general mineral land policy; but no discovery was made until 1909 when a valid discovery of oil was made by the locator. In the absence of any intervening location rendered valid by the discovery or by prosecution of diligent work, the discovery in 1909 would confer upon the locator the vested status of a true locator of mineral land. If the locator possessed that status in 1909 and thereafter conformed to and complied with the law, no act of the Government, short of proceedings in eminent domain, could deprive him of his right of property and he could not be subjected to the provisions of a subsequent withdrawal order of whatever source or authority.

United States v. McCutchen, 238 Fed., 575, p. 580.

DISTINCTION BETWEEN OIL AND PLACER LOCATIONS.

The act of 1897 gives a right to enter on lands that are "chiefly valuable" for oil. Section 2319 of the Revised Statutes recites that all valuable mineral deposits are open to exploration and purchase and the land in which they are found to location and purchase. As to lode and placer claims the right is given to explore and purchase "valuable mineral deposits" and the land in which they are found but with respect to oil, the right is given only to enter and obtain patent to lands which not only obtain oil, but which are "chiefly valuable therefor." In one case the value of the "deposits" is the criterion and in the other it is the value of the land, and this principle is not to be lost sight of in defining what will suffice for a discovery under the oil statute.

United States v. McCutchen, 238 Fed., 575, p. 583.

LIBERAL CONSTRUCTION OF STATUTE.

What constitutes discovery in its broad and comprehensive sense is the doing or the accomplishing of that thing with respect to the land sought to be appropriated which serves to impress upon it the quality of being land which is open to exploration or appropriation in the manner and pursuant to the law. Both judicial and departmental rulings have evinced a disposition to be liberal toward locators in the matter of the requirements as to what will suffice to constitute such discovery as to segregate the land sought to be selected from the public domain, and invest it with the attribute of mineral land, and subject it to private ownership or exploitation. If the locator has made such a location or a discovery upon certain defined land as to invest him with a right of property therein, he has made such location and discovery as to entitle him to a patent as against the Government. Conversely, if he has not made such discovery as to entitle him to a patent as against the Government, then he has made no such discovery as to vest him with rights in and to the property. The encountering by a locator in abandoned oil wells drilled by others of a small quantity of gas, both because of its inconsequence, nature, and extent, and also because of the fact that it was not, until after suit was brought, relied upon as a discovery validating the claim, can not, as a matter of law, authorize the court to adjudge that such encounter of gas sufficed to segregate the land in question from the unappropriated public domain prior to the withdrawal order of September 27, 1909.

United States v. McCutchen, 238 Fed., 575, pp. 584-593.

STATUTES RELATING TO MINING OPERATIONS.

CONSTRUCTION, VALIDITY, AND EFFECT.

COAL AND ORE MINES—EMPLOYMENT OF BOYS.

Section 1035 of the Alabama Code of 1907 applies to ore as well as coal mines but a distinction is made as to the prohibited employment of boys under 14 years of age in surface and underground mines, and the prohibition does not apply to an ore mine worked wholly from the surface.

Sloss-Sheffield Steel & Iron Co. v. Bearden, (Alabama), 74 Southern, 230, p. 231.

MEANING OF WORD "MINE."

The word "mine" is omitted from sections 1 and 2, Chapter 60, Acts of 1911 (West Virginia Code of 1913, secs. 530-531), amending Chapter 11 of the act of 1887, Chapter 15, Acts of 1891, and Chapter 75, Acts of 1905, as one of the places where minors of the prohibited age are not permitted to be employed, is not covered by any of the other places enumerated in the act and can not be restored by judicial construction.

Rhode v. J. B. B. Coal Co. (West Virginia), 90 Southeastern, 796, p. 797.

JUDICIAL NOTICE—PARTS OF MINE.

The courts of Kansas take judicial notice of the manner, largely regulated by statute, in which a coal mine in southeastern Kansas is operated under the shaft, entry, room-and-pillar plan or system and applying such matters in the construction of the statutes, the court must reach the following conclusions: Air courses are passages for conducting air. Entries are those places in coal mines used by the miners and other workmen generally in going to and from their work and through which coal is hauled from the necks of the rooms to the foot of the shaft. A room is a place in which a miner works and from which he mines coal. Traveling ways are places for the passage of workmen to and from different parts of a mine.

Ricardo v. Central Coal & Coke Co. (Kansas), 163 Pacific, 641, p. 643.

APPLICATION OF TERMS—QUESTION OF LAW.

The terms "room," "entry," "traveling way," and a number of others are used in the Kansas statute relating to the mining operations and must have definite and fixed meanings applicable in all situations where the shaft, entry, room-and-pillar system of mining is carried on. Accordingly when a place in a mine is definitely described and its relation to the other parts of the mine is fixed and certain, as

being a place in a room, an entry, an air passage, or a traveling way, it is a question of law for the court to determine whether such place is or is not in a traveling way.

Ricardo v. Central Coal & Coke Co. (Kansas), 163 Pacific, 641, p. 643.

DUTIES IMPOSED ON OPERATOR.

DUTY OF INSPECTION.

Whether the duty of inspection under the Virginia statute is imposed upon the mine foreman or boss is immaterial where a mine operator had knowledge of a squeeze in the mine and a boy of 17 years of age was put to work in that portion of the mine without an inspection by the mine foreman or boss, or by any other person on behalf of the operator. Under such circumstances the death of the boy, caused by a fall or rock due to an alleged squeeze, may be said to be the result of the operator's negligence, because of his failure to have the proper inspection made.

Big Vein Pocahontas Co. v. Repass, 238 Fed., 332, p. 335.

DELEGATION OF DUTY.

The mining statute of Virginia must be so construed as not to relieve a mine operator from seeing that all of its provisions are strictly complied with, nor from the duty imposed at common law to secure the reasonable safety of his employees and these duties are nonassignable, and if required by a foreman, boss, or fire boss, he shall be considered as acting for the mine operator as a vice principal.

Big Vein Pocahontas Co. v. Repass, 238 Fed., 332, p. 335.

DEFECTIVE CABLE—QUESTION OF FACT.

In the operation of a coal-cutting machine the question as to whether or not the cable extending from the machine to the jack pole and which served to hold the machine in place and which was subject to a strain of something like 4,000 pounds was worn and defective and whether a proper inspection of it would have disclosed its defective condition, can not be determined as a matter of law, but must be submitted to a jury as a question of fact.

Tennessee Coal, Iron & Railway Co. v. Wiggins (Alabama), 73 Southern, 516, p. 517.

EMPLOYMENT OF MINE FOREMAN—LIABILITY FOR NEGLIGENCE.

The statute of Tennessee (Acts 1903, ch. 237), places upon the mine operator the duty of employing a certified mine foreman. The failure of a mine operator to employ a mine foreman duly certified is immaterial in an action by a miner for damages for injuries where there is no evidence whatever that any of the duties required of the mine foreman were neglected.

Lively v. American Zinc Co. (Tennessee), 191 Southwestern, 975, p. 979.

DUTY TO ADOPT RULES.

The Kentucky statute (secs. 2738b-2738c) requires operators of coal mines to adopt rules for the safety of persons employed in and about the mines. A mine operator, who adopted rules, caused them to be printed in a book, and delivered a copy of the book to a miner, can not be held liable in damages for injuries to such miner received by him while violating a rule. It can not be objected that the rules were not adopted and promulgated in strict conformity to the statute for the reason that entirely independent of the statute, a mine operator has a right as an employer of labor to adopt reasonable rules and regulations for the safety and protection of his employees, and the violation of the rules so adopted and promulgated would be equally as effective to bar an action by an employee violating them as if the rules violated had been adopted and promulgated in accordance with the provisions of the statute.

Gatliff Coal Co. v. Peace (Kentucky), 192 Southwestern, 651, p. 653.

SAFE APPLIANCES.**MEANING OF SAFE APPLIANCES.**

The statute of Tennessee (Acts 1915, ch. 169) requires that hoisting machinery used for lowering miners into and out of a mine shaft be kept in a safe condition. This imposes the duty on a mine operator to make the appliances safe and not merely to exercise reasonable diligence to make them safe. When a term having a well-recognized meaning in common law is used in a statute that meaning will be given it in construing the statute, unless a different sense is apparent from the context, or from the general purpose of the statute. Where the word "safe" is closely followed by special provisions looking to safety, the statute will be construed to intend that the result shall be attained by the use of these means so far as they extend. The term "safe" when used in respect to appliances to be furnished by an employer to an employee means "reasonably safe," and "reasonably safe" means such tools as are in general use among employers of ordinary caution and prudence in the same line of business under the same circumstances.

Lively v. American Zinc Co. (Tennessee), 191 Southwestern, 975, p. 977.

CHANGE OF APPLIANCES NOT REQUIRED.

Under the statute of Tennessee, requiring the use of safety appliances and safety cages in lowering and hoisting miners in and out of a mine, a mine operator has the right to choose between different kinds and types of implements in general use, and is not subject to a charge of negligence, or a violation of the statute, because some other style than that adopted by him may be found safer. A mine operator

or other employer is not required to change his machinery in order to apply every new invention or superior improvement, and he can use appliances, even though they are shown to be less safe than other kinds in use for the same ends without being liable to his employees for the consequence of such use.

Lively v. American Zinc Co. (Tennessee), 191 Southwestern, 975, p. 979.

DUTY TO COMPLY WITH PROVISIONS OF STATUTE.

The statute of Tennessee (Acts 1915, ch. 169) requires that the elevator in a mine shaft used for lowering or hoisting persons shall be fitted with an improved safety catch and a sufficient metal covering overhead. It also requires sufficient flanges to be attached to the sides of every drum and other machinery used for lowering and hoisting persons with adequate brakes attached to the drum. The duty to comply with these and similarly specific provisions of the statute, designed as precautions against accidents and injuries is absolute and they can not be satisfied by a near approximation or by the exercise of reasonable diligence or ordinary care in an effort to comply.

Lively v. American Zinc Co. (Tennessee), 191 Southwestern, 975, p. 977.

EMPLOYMENT OF BOYS.

VIOLATION—EMPLOYMENT OF CHILD OF PROHIBITED AGE.

Sections 71 and 72 of chapter 15H of the West Virginia Code of 1913 prohibiting the employment of a minor under 14 years of age impliedly permits the employment of minors over that age, and minors over that age are comprehended in the definition of employee protected and bound by the provisions of the workmen's compensation act.

Rhode v. J. B. B. Coal Co. (W. Va.), 90 Southeastern, 796, p. 798.

VIOLATION OF DUTY—EMPLOYMENT OF BOY—NEGLIGENCE.

A coal-mine operator who employed a boy under 16 years of age to work in his coal mine in violation of the statute is liable aside from any question as to the prohibited age, where the operator placed the boy, without giving him any instructions at a trap door to open and close the same and where a driver coming from the opposite direction without waiting for the boy to open the door ran his motor through the closed door and ran over the boy, who was in the entry on the opposite side and close to the door.

Carter Coal Co. v. Love (Kentucky), 190 Southwestern, 481, p. 482.

EMPLOYMENT OF CHILDREN—LIABILITY.

The statute of Kentucky (sec. 331a, subsec. 9) prohibits coal-mine operators from employing children under 16 years of age in any

capacity in or about, or in connection with any mine, coke oven, or quarry. In an action against a mine operator for the death of a boy under 16 years of age employed by an operator in violation of this statute, it is sufficient to sustain a recovery to prove that the boy was employed by the defendant operator and that he was under 16 years of age.

Carter Coal Co. v. Love (Kentucky), 190 Southwestern, 481, p. 482.

EMPLOYMENT OF BOY—RIGHT OF FATHER TO RECOVER.

In an action against a coal-mine operator for the death of a boy under 16 years of age, the fact that the father, the sole beneficiary and for whose use the suit was brought, consented to the employment of the boy, can not be taken advantage of on the trial of the case by the mine operator in the absence of an affirmative answer pleading the same.

Carter Coal Co. v. Love (Kentucky), 190 Southwestern, 481, p. 483.

VIOLATION BY MINER.

INTOXICATED MINER ENTERING MINE—PROXIMATE CAUSE OF INJURY.

The statute making it an offense for any person to enter a mine under the influence of intoxicating liquors or to carry intoxicating liquors into a mine (Alabama Laws, 1911, p. 536) was enacted for the benefit of the mine owner or operator and of his employees as distinguished from the public generally. Ordinarily the violation of a statute is negligence per se; but the violation of a statute to constitute negligence that renders a mine operator liable or will defeat a miner's action for damages must be the proximate cause of the injury complained of. A miner entering a mine under the influence of intoxicating liquors is guilty of an offense against the statute and such intoxication is a defense to an action by the miner for injuries where it is made to appear that such intoxication was the proximate cause of the injury complained of.

Reynolds v. Woodward Iron Co. (Alabama), 74 Southern, 360, p. 362.

MINER'S WORKING PLACE.

EXTENT—TRAVELING WAY.

The whole of a room in which a coal miner works while mining coal and while pushing cars used by him in his work is his working place, and no part of it is a traveling way within the meaning of section 6276 of the Kansas General Statutes of 1915.

Ricardo v. Central Coal & Coke Co. (Kansas), 163 Pacific, 641, p. 643.

EFFECT ON CONTRIBUTORY NEGLIGENCE.**MINER'S KNOWLEDGE OF DEFECTS—FAILURE TO INFORM OPERATOR.**

The proviso of section 3910 of the employers' liability act of Alabama (Alabama Code, 1907), relieving an employer or operator from liability where the employee or miner knew of the defect and failed to inform the employer or operator, is intended to relieve the employee or miner from the imputation of contributory negligence predicated on the fact of his remaining in service after knowledge of the defect or negligence in an action by him for injuries; but it does not relieve him of the duty to give information of such defect or negligence when he knows of it and the employer or operator has no knowledge or notice thereof. But the employee or miner is not required to inform the employer or operator of the defect or negligence where the employer or operator already has knowledge of such defect or negligence.

Reynolds v. Woodward Iron Co. (Alabama), 74 Southern, 360, p. 362.

AUTHORITY OF MINE FOREMAN.**AUTHORITY OF MINE FOREMAN TO EMPLOY ASSISTANT.**

Section 2726 (Kentucky statute) provides that the mine foreman shall have charge over the inside workings of a mine and of the persons employed therein. It also provides that assistants to the mine foreman may be employed by the operator or superintendent. Under this statute a mine foreman has no authority to employ an assistant mine foreman. But if a mine foreman employs an assistant and the assistant with the knowledge and acquiescence of the superintendent or operator of the mine acts in that capacity, their knowledge and acquiescence in the employment by the mine foreman and what the assistant was doing, would constitute an approval or a ratification of the employment and would place the assistant in the attitude as if he had been employed by the operator or mine superintendent as required by the statute. Under such circumstances a mine operator would be liable for the acts of such assistant to the same extent as if he had been appointed or employed by the operator or superintendent. But such an assistant can not recover for an injury received by a premature shot made by a miner in a part of the mine distant from the place where such assistant was required to perform his ordinary duties as a regular employee. If he was not an authorized assistant mine foreman, he was purely a volunteer and had no business to leave his work and go to another part of the mine where the shot was fired. If he went as assistant mine foreman, the mine operator would not be liable for injuries received by him from the shot. If he went as a volunteer, he can not recover, because as a volunteer he had no

business at the place where the shot was fired and no duty to perform there.

Harris v. Lamb Coal Co. (Kentucky), 190 Southwestern, 121, p. 123.

MINERS' WASH-ROOM LAW.

MINERS' WASH-ROOM LAW—DUTY TO FURNISH SUPPLIES.

The miners' wash-room law of Indiana (Burns Stat. 1914, sec. 8623) requires miners' wash rooms to be supplied with cold and warm water and to be provided with all necessary facilities for persons to wash, and with lockers for the safe-keeping of clothing and relieves the operator from furnishing soap or towels. The proviso which excuses the mine owner from furnishing soap and towels makes it clear, from the other requirements that all other things essential to the equipment and maintenance of the wash room shall be furnished by the mine owner on the principle that where one thing is mentioned in a statute it is to the exclusion of all other things. Where the statute excepts from the things to be furnished in the maintenance of the wash-room soap and towels, it follows that all things necessary to its proper equipment and maintenance shall be furnished by the operator.

Princeton Coal Co. v. Fettinger (Indiana), 114 Northeastern, 406, p. 407.

MINERS' WASH-ROOM LAW—VALIDITY.

The miners' wash-room law of Indiana (Burns Stat. 1914, sec. 8623) requiring that miners' wash rooms shall be maintained by the coal companies is no more in conflict with the constitution than the part requiring that the wash room should be built by the mine owner. The act is constitutional and valid.

Princeton Coal Co. v. Fettinger (Indiana), 114 Northeastern, 406, p. 407.

FAILURE TO FURNISH WASHHOUSE—LIABILITY OF SUPERINTENDENT.

The statute of Indiana requiring mine operators to erect and maintain washhouses for the benefit of the employees (Burns Stat. 1914, sec. 8623) includes mine superintendents within its operation, and a mine superintendent who fails, on request of the miners to provide a wash room, is liable to a criminal prosecution under the statute.

McQuade v. State (Indiana), 115 Northeastern, 583.

WORKMEN'S COMPENSATION ACT.

APPEAL—QUESTIONS CONSIDERED.

On appeal from a decision of the industrial board provided for by the workmen's compensation act of Illinois (Laws 1913, p. 347) the court, in reviewing the proceedings of the board, can only review

questions of law and can only determine from the facts recited in the decision of the industrial board whether that body acted within its powers in making the award.

Kerens-Donnewald Coal Co. v. Industrial Board, 115 Northeastern, 225, p. 227.

CONCLUSIVENESS OF FINDINGS.

Under section 19 of the workmen's compensation act of Illinois (Laws 1913, p. 347) when the industrial board acts within its power, its findings upon the facts are conclusive upon the courts.

Illinois Midland Coal Co. v. Industrial Board, 115 Northeastern, 527, p. 528.

FINDING—TOTAL AND PERMANENT DISABILITY.

A finding of the industrial board under the Illinois workmen's compensation act (Laws 1913, p. 347) that the injured workman is "now totally disabled," is not a finding and does not have the effect of a finding that the injured workman is permanently disabled. A man might now be totally disabled and yet not be permanently disabled.

Illinois Midland Coal Co. v. Industrial Board, 115 Northeastern, 527, p. 528.

JURISDICTION OF INDUSTRIAL BOARD.

The jurisdiction of the industrial board to review the proceedings of the committee of arbitration does not depend on the filing of a stenographic report within the time specified by the statute. It is sufficient if at any time before the hearing a properly authenticated stenographic report is filed with the industrial board. The date fixed by the statute as to the filing of the stenographic report is directory and not mandatory, and is not jurisdictional.

Illinois Midland Coal Co. v. Industrial Board, 115 Northeastern, 527, p. 528.

PRACTICE AND PROCEDURE.

The workmen's compensation act of Illinois (Laws 1913, p. 347) provides (sec. 16) that the industrial board may make rules and orders for carrying out the duties imposed upon it, and that the process and procedure shall be as simple and summary as reasonably may be. It provides also (sec. 19) for the appointment of a committee of arbitration, where the employer and employee can agree. There is no time fixed within which the committee shall make its investigation and report, and the statute in no way intimates that the committee of arbitration can be relieved from making a report because the members do not agree, but it positively provides that it shall report. When a committee was duly appointed and reported that it could not agree, it was the duty of the board to refer the matter back and require a final report from the committee.

Kerens-Donnewald Coal Co. v. Industrial Board, 115 Northeastern, 225, p. 226.

MINES AND MINING OPERATIONS.

NEGLIGENCE OF OPERATOR.

MINER'S KNOWLEDGE OF DANGER.

The fact that a foreman directed a rope tender to disentangle with his hands the rope while in motion in case it caught under the cross-ties and that in following the directions the miner was injured is not sufficient to charge the mine operator with negligence where presumptively the miner was a man in full possession of all of his faculties and fully appreciated the danger of taking hold of the rope while it was in motion.

Victor American Fuel Co. v. Eidson, 237 Fed., 999, p. 1102.

INCREASING DANGER.

A mine operator is liable on the ground of negligence for the death of a miner who with others and with the knowledge and consent of the operator entered a mine to extinguish a fire and while so in the mine seeking to extinguish the fire and without any knowledge on his part, the operator caused the fan to be started thereby driving the smoke and heat into the entries and places where the miner was compelled to go, causing his suffocation and death.

Western Coal & Mining Co. v. McCallum, 237 Fed., 1103, p. 1105.

AGREEMENT TO HAUL MINERS—CONTRACTING AGAINST NEGLIGENCE.

A railroad company constructed and operated a branch line from its main road to certain coal mines in operation and shipping over its road. By agreement between the mine operator and the miners on the one part, and the mine operator and the railroad company on the other, it was agreed that the railroad company should haul the miners from the mine to their homes for \$1.50 per month and that this amount should be deducted from their wages by the operator and by him paid to the railroad company monthly. The carriage was to be over the line of the railroad company and in its cars and subject to its direction and control. Under such circumstances each miner while being transported was a passenger on the road of the railroad company, and the company was, under the circumstances, a common carrier and it can not, as a common carrier, avoid responsibility for its own negligence, resulting in an injury to one of the coal miners, and can not by contract relieve or limit its liability as such common carrier.

Vandalia Railway Co. v. Stevens (Indiana App.), 114 Northeastern, 1001, p. 1006.

METHOD OF MOVING CARS.

An employee of a coal-mine operator was given the duty of moving the loaded cars away from the tippie and setting empties at the tippie to be loaded. Another employee assisting in the work was directed to release the brakes on two empty cars and start them down the track to be placed at the tippie. As the cars came down of their own momentum they ran against another car standing on the track and pushed it against another car a few feet distant. In some manner unknown the first employee was standing between the couplings of these two cars, or passing between them as the first car was struck by the two sent down, and instantly killed. The mining company can not be charged with negligence where there is nothing to show the purpose for which the deceased employee passed between the cars at the particular time and where it also appeared that the empty cars started down to the tippie went at a speed of from 2 to 6 miles an hour, and where it appeared that there was nothing unusual in the manner of placing the empty cars and that the deceased employee was familiar with the methods adopted in placing the empty cars at the tippie, and nothing to show that it was customary to give any notice of the movement of cars down the incline, or to have a person stand on the front end for the purpose of warning persons who might happen to be on or near the track, or that the deceased employee had any right to depend on any warning or signal of the movements of the cars; but on the contrary, it appeared that the work in which he was engaged required him in the exercise of ordinary care for his own safety to keep a lookout for the movement of empty cars coming down the incline.

West Kentucky Coal Co. v. Heady (Kentucky), 190 Southwestern, 475, p. 477.

PROOF OF CUSTOM AS AFFECTING LIABILITY.

Evidence of a custom in a mining district that a mine operator should look after the safety of roofs of entries to the mine and of the operator's responsibility for the condition of such roofs when notified of their defects and that an entry means a passageway high enough for mules to pull small cars through is admissible in an action for personal injuries resulting from the falling of a roof, where the miner was engaged in the entry.

Ricardo v. Central Coal & Coke Co. (Kansas), 163 Pacific, 641, p. 644.

INJURY RESULTING FROM ONE OF TWO CAUSES.

Where an injury resulted to a miner from one of two causes for one of which and not the other the mine operator would be liable, the burden rests upon the injured miner to show with reasonable

certainty that the cause for which the operator is liable produced the injury, and if the evidence leaves it to conjecture the injured miner must fail in his action.

Graves v. United States Smelting Co. (Missouri App.), 192 Southwestern, 472, p. 473.

UNGUARDED TROLLEY WIRE—QUESTION OF FACT.

A coal-mining corporation operated its cars in its coal mines by means of electricity. The main trolley wire was strung in the heading 4 feet 7½ inches above the rail and about 6 inches outside of the rail. The wire was at all times heavily charged. At the different rooms extending off from the heading were switches by which the miners could take the cars into their several rooms. In order to get a car from the main track onto the switch it was necessary for a miner to go from his room into the heading and "slew" or shift the car from the main track to start onto the switch. The front end of the car was first shifted and the miner must then go to the other end and shift or slew it and then push the car into his room. A miner desiring a car in his room went into the heading, met a car, slewed or shifted the front end as usual, and in attempting to pass from the front to the rear, he stepped on the side of the car on which the trolley wire was strung and came in contact with the heavily charged wire and received a shock causing his death. The miner knew of the presence of the trolley wire, but had nothing whatever to do with placing it in position or with maintaining it in a dangerous or safe condition. The usual custom of the miners in shifting or slewing the car was to pass on the side opposite from the trolley wire. Under such circumstances and in view of the fact that many other mine operators in the community maintained unguarded wires in the operation of their cars by electricity, the questions of negligence of the operator as well as the contributory negligence of the miner are questions of fact to be determined by the jury.

Reynold v. Woodward Iron Co. (Alabama), 74 Southern, 360, p. 361.

PROOF OF NEGLIGENCE.

Where a miner was injured in the course of his employment the burden is upon him to show that his injury was caused by the negligence of the mine operator where his action for damages is based upon the alleged negligence of the operator.

Western Coal & Mining Co. v. Harrison (Arkansas), 192 Southwestern, 190, p. 191.

PROOF OF NEGLIGENCE.

In an action by a miner against a mine operator to recover for injuries received while in the service of the operator, the miner must establish three things to entitle him to a recovery: (1) Negligence

on behalf of the mine operator. (2) The happening of the accident resulting in the injury complained of. (3) That the injury complained of is the proximate result of the accident and the negligence of the defendant.

North Jellico Coal Co. v. Stewart (Kentucky), 191 Southwestern, 451, p. 453.

PERSON INVITED ON PREMISES BY FOREMAN—INJURY.

The chief engineer of a mining company after an electrical storm found an employee lying in an unconscious and dying condition as the result of an accident. In the absence of all superior officers, he sent a messenger for a physician. A physician obeyed the summons and on his arrival examined the body of the unconscious employee and went to the telephone to call an assistant. The physician in attempting to use the telephone was himself severely injured by a shock due to the negligent construction of the telephone lines, whereby a power wire carrying a high voltage had been strung immediately over the telephone wires and during the electrical storm the power wire had become displaced or broken, falling upon the telephone wires, charging them and causing the injuries to the employee and to the physician. Under the circumstances and in the absence of the company's chief officers and of the danger in attempting to call them by telephone, the chief engineer was justified in dispatching a messenger for a physician, and the physician on his arrival was not a trespasser but an invitee on the premises and it was the operator's duty to keep the premises in a safe condition, that he might not be injured while he was there professionally in response to the invitation. It is not a question of the authority of the chief engineer to bind the operator for the value of the services rendered by the physician to the injured employee, but he had authority to invite the physician upon the premises for the purpose of rendering aid to the injured employee, and the physician went upon the premises by express invitation and not as a licensee. The physician did not exceed his invitation by attempting to use the telephone. He needed assistance and the telephone being within reach, he did what anyone under like circumstances would have done. The telephone was one of the means he attempted to use to carry out the purpose of the invitation. The negligence in the defective construction of the power and telephone lines, and the failure to provide safeguards to prevent the power wire from falling upon the telephone wire is sufficient to render the operator liable, although the operator did not have actual knowledge that the power wire had come in contact with the telephone wire.

Jones v. Pennsylvania Coal & Coke Corporation (Pennsylvania), 99 Atlantic, 1008, p. 1009.

NEGLIGENT ORDERS OF SUPERINTENDENT—QUESTION OF FACT.

In an action by a helper in the operation of a coal-cutting machine for damages for injuries received by being caught by a defective cable and dragged into the teeth of the machine, the question as to whether or not the order of the superintendent in ordering the helper to set up the jack pole and to hold the same in position when being drawn out of position by the force or strain on the cable attached to the jack pole was a negligent order, is a question of fact to be determined by the jury.

Tennessee Coal, Iron & Railway Co. v. Wiggins (Alabama), 73 Southern, 516, p. 518.

DUTY OF OPERATOR TO FURNISH SAFE PLACE.**SAFE PLACE—QUESTION OF FACT.**

The duty rests upon a mine operator to use reasonable care to furnish a miner a safe working place; but the question as to whether or not this duty was performed and as to whether or not the place was safe is one for the determination of the jury.

Big Vein Pocahontas Co. v. Repass, 238 Fed., 332, p. 334.

WHAT CONSTITUTES SAFE PLACE.

The doctrine of safe place does not apply to a place that is not in fact dangerous or unsafe, nor to a place where it appears that the work at such particular place was attended with more difficulty than other places. The doctrine does not apply where a laborer in a coal mine was employed to push cars through a tunnel and in consequence of a fall of slate the track had been raised, making an incline over the fall, and in attempting to push his car over the place, his strength being insufficient, the car rolled back upon him, causing the injury complained of.

Saulsbury v. Elk Horn Consolidated Coal & Coke Co. (Kentucky), 192 Southwestern, 20, p. 21.

SAFE PLACE—APPLICATION OF DOCTRINE.

The rule requiring a mine operator to furnish a safe place for his miners to work does not apply to a miner who himself is making an unsafe place safe.

Graves v. United States Smelting Co. (Missouri App.), 192 Southwestern, 472.

The rule that a mine operator or other employer must exercise reasonable care to furnish a miner or an employee with a safe place in which to work does not apply where the miner or employee is himself creating the place in which he works, or where the danger was such as was created by the miner or employee in the progress of his work.

New Hughes Jellico Coal Co. v. Gray (Kentucky) 191 Southwestern, 78, p. 79.

OPERATOR'S PROMISE TO REPAIR.**PROMISE TO REPAIR—RELIANCE.**

Where a miner complains of the dangerous condition in which he had to work that is due to the operator's negligence and the operator promises to remedy the same, the miner may in reliance upon the promise remain for a reasonable time in the employment without assuming the risk or depriving himself of the right to recover for injuries received because of such dangerous condition.

New Hughes Jellico Co. v. Gray (Kentucky), 191 Southwestern, 78, p. 79.

A miner working at the bottom of a shaft complained to the operator of an extra and unusual amount of coal falling down the shaft and the danger thereof caused by constructing and maintaining the dumping blocks too low by which self-dumping cages were unloaded at the top of the shaft. On the promise of the operator to remedy the difficulty and remove the danger the miner had a right to rely for a reasonable length of time on the promise to repair and remedy the danger, and it is proper, under certain circumstances, for a court to submit to a jury the question of whether or not a reasonable time for making the repairs had expired.

Western Coal & Mining Co. v. Harrison (Arkansas), 192 Southwestern, 190, p. 191.

PROMISE TO REMEDY—MINER CONTINUING WORK.

The rule that a miner may continue to work in a dangerous place for a reasonable length of time after the operator has promised to remedy and remove the danger, without subjecting himself to the charge of contributory negligence, does not apply where the risk incurred in remaining in the place is so great and imminent that a reasonably prudent man would not incur it. The promise of the operator to remedy the conditions does not make it an insurer of the safety of the place for a reasonable time thereafter. The miner must still exercise ordinary care for his own safety, and if he exposes himself to dangers that are so threatening or obvious as likely to cause injury at any moment, he is, notwithstanding the promise of the operator, guilty of contributory negligence and can not recover for an injury received while continuing in such place.

New Hughes Jellico Co. v. Gray (Kentucky), 191 Southwestern, 78, p. 79.

Consolidated Coal Co. v. Spradlin (Kentucky), 190 Southwestern, 1069.

Saulsbury v. Elk Horn Consolidated Coal & Coke Co. (Kentucky), 192 Southwestern, 20.

Western Coal & Mining Co. v. Harrison (Arkansas), 192 Southwestern, 191, p. 192.

MINER'S WORKING PLACE—SAFE PLACE.**MINER MAKING SAFE PLACE.**

The doctrine of safe place and of fellow servant and of assumed risk have no application where an experienced miner or timberman is set to work in a compartment to remedy defects and to make the compartment safe for miners using a ladderway in going into and coming out of the mine.

Drajicevic v. Champion Copper Co. (Michigan), 160 Northwestern, 451, p. 452.

MINER'S ROOM AS WORKING PLACE.

In an action by a miner for damages for injuries caused by a fall of rock from the roof, it appeared that the miner was sent into a room that had been partly worked out by another miner; that on the day before the miner went to work he examined the roof and concluded that the roof was bad and on complaint the mine boss promised to send in props and timbermen to fix the roof. Before the miner went to work he found some props and set them so that he could work in the face of the coal. The injury complained of was caused by a rock which fell from the roof in that part of the room from which the coal had been taken before he began work. Evidence was admitted tending to show that it was the custom in the mine for the mine operator to put a room in a safe condition when starting a miner to work in it, after it had been partly worked out by another miner. If any part of the miner's room was a traveling way under the statute, proof of such custom would be unnecessary and wholly immaterial. The existence of the custom argues that the whole of the room is the miner's working place and that no part of it is a traveling way. The evidence shows that the miner received his cars on the switch at the entry to his room, pushed the cars in, loaded them and pushed them out again to the switch, showing that the miner used the whole length of the room as his working place. Under such circumstances a court is compelled to say that the place in which the plaintiff was injured was his working place and not a traveling way within the meaning of the statute. A mine operator is authorized under the statute to require a miner to examine and prop his working place; and if a miner is so instructed and fails to make his working place safe, he can not recover from the operator for injuries due to his failure to make his working place safe.

Ricardo v. Central Coal & Coke Co. (Kansas), 163 Pacific, 641, p. 644.

DUTY OF MINER TO INSPECT.

Under a custom existing in a mine each miner was required to observe and to inspect the condition of his room in which he worked

and also the room neck where he was required to do at least a part of the work, and if he discover any loose slate or stone to remove it if possible and if not to report it to the mine boss. This custom did not impose the exclusive duty of inspection upon the miner, where it is shown that there was a mine boss or inspector and an assistant whose duty it was to inspect the condition of all parts of the mine where miners were at work. The inspector or his assistant visited the miner's room daily and made daily inspection of the room neck. The injured miner never discovered or reported any loose slate to the mine boss and the piece of slate that fell and injured the miner was the only one that ever fell from the room neck. Under the circumstances the mine operator can not be charged with negligence.

North Jellico Coal Co. v. Stewart (Kentucky), 191 Southwestern, 451, p. 453.

DUTY TO WARN OR INSTRUCT.

DUTY TO WARN.

To order an employee to perform a dangerous service, standing alone, is not negligence, and to allege such fact without alleging other facts showing the duty on the part of the mine operator to warn, and a violation of such duty, does not charge negligence. While all machinery is more or less dangerous, still the duty to warn exists only in special cases. From the allegations of the complaint, it must be presumed that the plaintiff was a man in the full possession of all his faculties and was capable of appreciating and knowing the dangers of the service he was ordered to perform as well as the foreman; therefore the allegation that he did not know of the danger does not help his case.

Victor American Fuel Co. v. Edison, 237 Fed., 999, p. 1102.

DUTY OWING TO A VOLUNTEER.

A mechanic and miner who with the knowledge and consent of the mine foreman with others entered a mine for the purpose of putting out a mine fire, though classed under the term "volunteer" used in its broadest sense, assumed the risk of the conditions as they existed when he entered the mine, but he did not assume a new risk when the operator by affirmative action increased the danger. Under such circumstances it can not be said that the operator owed him no duty or that the duty owing consists merely in not intentionally injuring him. An employee who makes himself useful in a matter not covered by an express command, but whose services are accepted, though not in any express words, does not, as a matter of law, put himself outside the limits of his employment.

Western Coal & Mining Co. v. McCallum, 237 Fed., 1103, p. 1105.

CONTRIBUTORY NEGLIGENCE OF MINER.**CONTRIBUTORY NEGLIGENCE AS QUESTION OF LAW.**

A miner working in his room discovered a crack in the roof and a piece of slate resembling a horse back that extended beyond and above the face of the coal. Before removing the coal, he reported the conditions as sufficiently dangerous to request the foreman to timber. However, he and his buddie continued working and removed the coal from beneath the slate so extending beyond the face of the coal. As an experienced miner, he knew that if the coal was removed from beneath the horse back, it would fall if not timbered, and he knew that it was not timbered. After removing the coal beneath the horse back he got upon the bottom bench and began shearing the rib, with knowledge of the fact that the slate was likely to fall and injure him at any time. Notwithstanding the foreman's promise to timber, the miner voluntarily exposed himself to an obvious, known, and imminent danger, which he knew was likely to cause injury at any moment, and he was therefore guilty of contributory negligence as a matter of law.

New Hughes Jellico Coal Co. v. Gray (Kentucky), 191 Southwestern, 78, p. 79.

BURDEN OF PROVING CONTRIBUTORY NEGLIGENCE.

In hauling coal cars out of a mine it was necessary at an incline to sprag the wheels of the car, and when the car reached the bottom of the incline it was customary for the driver to jump from the cars to protect himself. In an action for the death of a car driver caused by his failure to jump from his car, where the evidence established primary negligence on the part of the mine operator which proximately caused the injury complained of with nothing in the circumstances establishing contributory negligence on the part of the car driver, and where it did not appear that the deceased driver knew of the necessity of jumping from the car, the burden of proving the contributory negligence of the car driver rested upon the mine operator when it was interposed as a defense to the action.

Folsom-Morris Coal Co. v. Dillon (Oklahoma), 162 Pacific, 696.

DUTY OF MACHINE OPERATOR—WARNING TO HELPER.

In an action by a machine helper for damages for injuries caused by being caught by the cable extending from the machine to the jack pole and drawn into the knives or bits of the machine, causing the injury complained of, it is proper, on the cross-examination of the machine operator to whose orders the helper was subject, to ask, and to require him to state, if it was not the purpose and custom in operating the machine to notify the helper whose duty it was to hold

the jack pole in place, when the machine was started and when the bits were put into the coal, in order that he might get out of the way. The evidence is proper in such a case on the question of the contributory negligence of the injured helper, as a failure of the machine operator to give the helper notice of the starting of the machine would tend to free him from the charge of contributory negligence.

Tennessee Coal, Iron & Railway Co. v. Wiggins (Alabama), 73 Southern, 516, p. 518.

INSTRUCTION—BURDEN OF PROOF.

In an action for damages for the death of a car driver killed while driving his first trip of cars in a coal mine, where the mine operator pleads as a defense that the plaintiff was guilty of contributory negligence, and where the plaintiff's evidence shows that the operator was guilty of negligence, but with nothing in the circumstances establishing contributory negligence on the part of the plaintiff, the court must instruct the jury that the burden of proving contributory negligence under such circumstances is upon the mine operator.

Folsom-Morris Coal Co. v. Dillon (Oklahoma), 162 Pacific, 696.

FAILURE TO OBEY INSTRUCTIONS.

A miner reported that the roof of his working place was bad and the bank boss instructed him to the effect that he did not want to shoot the roof down unless it was necessary, but if it did get bad, to go ahead and shoot it. This instruction gave the miner not only authority, but directions, to shoot down the roof whenever, in his judgment, it became necessary, and put upon him the duty of inspection to ascertain its condition as the work progressed. Under such circumstances the miner was guilty of such contributory negligence in failing to obey the directions of the mine boss as would defeat his right of action to recover damages for injuries sustained by reason of such disobedience.

Consolidated Coal Co. v. Spradlin (Kentucky), 191 Southwestern, 78, p. 79, January, 1917.

Graves v. United States Smelting Co. (Missouri App.), 192 Southwestern, 472.

KNOWLEDGE OF DANGER.

A laborer was employed in a mine to push the cars of coal over a track and through a tunnel to a dumping place. There had been a fall of slate in the tunnel and the track over which the cars were pushed was laid over the pile of slate, making a hump in the track. The employee had knowledge of the situation and of the condition of the track. In attempting to push a load of coal over the hump his strength was insufficient and the car started back and ran against and upon him, causing the injury complained of. Under the circum-

stances and with knowledge of the condition of the track he was held to be guilty of such contributory negligence as to prevent a recovery.

Saulsbury v. Elk Horn Consolidated Coal & Coke Co. (Kentucky), 192 Southwestern, 20, p. 21.

VIOLATION OF RULES BY MINER—RECOVERY.

A coal-mine operator has a right to adopt reasonable rules and regulations for the safety and protection of his miners and employees, and when rules of this nature have been adopted an employee or miner who is familiar with the rules can not recover from the operator damages for injuries received at a time when he was violating the rules, or for injuries that would not have been received except for his violation of the rules.

Gatliff Coal Co. v. Peace (Kentucky), 192 Southwestern, 651, p. 653.

ASSUMPTION OF RISK.

RISKS ASSUMED.

VOLUNTARY SERVICE.

A machine foreman working in a mine and pursuing the duties required of him on hearing a shot fired in another part of the mine that he thought was fired out of order or not according to the schedule for firing shots, left his work and went to the part of the mine and entered the room where he thought the shot had been fired. As he entered the room another shot that had been prepared exploded causing the injury complained of. In going to the place where he received the injuries, he was not acting in his capacity as machine foreman or looking after the machine men as they had nothing to do with shooting or blasting coal. He did not receive his injuries while working in his capacity as machine foreman or while doing anything connected with his duties as such. His duty as machine foreman did not require him to give any attention to shots fired out of time or to go to the place where the noise from an explosion came from, or to investigate the cause of it. It follows that he could not recover damages for injuries sustained at a time when he was in a place where his duties did not require him to be and while acting purely as a volunteer.

Harris v. Lamb Coal Co. (Kentucky), 190 Southwestern, 121, p. 122.

ORDINARY RISKS OF EMPLOYMENT.

An employee in a coal mine assumes not only such risks as from the nature of the business, as ordinarily conducted, he must have known, but also those which the exercise of his opportunities for inspection would have disclosed to him.

El Paso Coal, Land & Fuel Co. v. Howell (Colorado), 161 Pacific, 293, p. 294.

MINER MAKING PLACE SAFE.

An experienced timberman in a mine, 32 years of age, engaged in repairing a compartment in a mine shaft, which because of its age was in need of repair so as to make it a safer place for the miners who had occasion to use the ladderway in this compartment can not recover from the mine operator for injuries caused by falling rock in the process of the work.

Drajicevic v. Champion Copper Co. (Michigan), 160 Northwestern, 451.

KNOWLEDGE IMPLIED.

An employee in a coal mine assumes the ordinary risks incident to the employment, although he does not have actual knowledge of the risk, if it is such that an ordinarily prudent man, under the circumstances, could by reasonable diligence have discovered it.

El Paso Coal, Land & Fuel Co. v. Howell (Colorado), 161 Pacific, 293, p. 294.

INJURY FROM CURTAIN—QUESTIONS OF FACT.

A coal-mine operator caused to be suspended at the entrance to one of the rooms in a coal mine a curtain of excessive and unnecessary length so that a part of it lay upon the floor of the mine and upon the track over which the coal was hauled out. A driver of coal trips was required to stop his car at the entry and pass through the curtain in order to stop his train and attach another car, and while alighting from his car for such purpose, owing to the darkness of the entry, he was tripped by stepping on that part of the curtain which lapped and folded on the floor of the entry and was thrown down, receiving the injuries for which he sued. The complainant had no knowledge of the curtain's excessive length and to charge him with the assumption of the risk would involve the question whether or not an ordinarily prudent man would under the particular circumstances have noticed that the curtain was too long and this is a question of fact that must be determined by the jury upon all the facts and circumstances of the case.

El Paso Coal, Land & Fuel Co. v. Howell (Colorado), 161 Pacific, 293, p. 294.

INJURY TO ASSISTANT MINE FOREMAN—VOLUNTEER.

A coal-mine operator can not be charged with negligence and rendered liable in an action by a person professing to act as assistant mine foreman, or otherwise as a volunteer, where the operator had fixed a regular time for firing shots by the miners, and where a miner mistaking the time had prematurely prepared his shots and where on the firing of the first shot before the fixed time, the assistant mine foreman left his place of work and went to the part of the mine where the shot was fired and where, as he entered the room, a second shot went off, causing the injuries complained of. If the employment as

assistant mine foreman was regular and the injured person was in fact an assistant mine foreman, the internal operations of the mine were under his directions; but if he was not regularly employed as assistant mine foreman, he was then a volunteer, not in the course of his employment, at the time of the injury, and in neither event could he recover where there was no negligence on the part of the mine operator with reference to the shots.

Harris v. Lamb Coal Co. (Kentucky), 190 Southwestern, 121, p. 123.

RISKS NOT ASSUMED.

FAILURE TO PROVIDE SAFE PLACE.

A mine operator may be guilty of such negligence as will render him liable for an injury to a miner caused by blocks or pieces of coal falling down the shaft, where in the operation of self-dumping cages the dumping blocks are made and maintained too low, as this causes the coal to lodge on the edge of the chute and as the cage would swing back it would knock the coal into the shaft. Miners working at the bottom of a shaft and the bottom cager assume the risk of pieces of coal falling down the shaft due to the overloading of cages and the method of operating cages, yet the miners at the bottom of the shaft do not assume the risk of large and unusual amounts of coal falling down the shaft.

Western Coal & Mining Co. v. Harrison (Arkansas), 192 Southwestern, 190, p. 191.

DANGERS FROM CURTAIN AT ENTRY.

A mine operator hung at the entry a curtain longer than necessary so that a part of it lay upon the floor and on the tracks where coal was hauled out. A driver of coal cars was required to stop his trip at the curtain in order to pass through and to take on another car. The curtain made the entry dark and in alighting from his car the driver, without knowledge of the excessive length of the curtain, and of the fact that a part of it lay upon the floor and across the track, caught his foot in the folds of the curtain causing him to stumble and fall and thereby received the injuries for which he complained. In the light of the circumstances it can not be said as a matter of law that the knowledge of the curtain's excessive length should be imputed to the driver, or that having such knowledge there was danger to him in such excessive length of the curtain. The danger was not obvious and reasonable men might draw different conclusions as to the complainant's duty to recognize that injury to him might result from the length of the curtain. Accordingly, it can not be said as a matter of law that the complainant assumed the risk incident to the excessive length of the curtain.

El Paso Coal, Land & Fuel Co. v. Howell (Colorado), 161 Pacific, 293, p. 295.

NEGLIGENCE OF FELLOW SERVANT.**LIABILITY OF OPERATOR.**

A miner was injured by being struck in the eye with particles of rock and dirt caused by a fellow miner striking a bowlder with a hammer. There was no negligence on the part of the mine operator and the fellow miner in striking the bowlder was engaged in the ordinary duties of the employment as was the miner who received the injury. Under such circumstances there can be no recovery, it being caused by the act of a fellow servant.

Misenhelter v. Geronimo Lead & Zinc Co. (Missouri App.), 192 Southwestern, 147, p. 148.

VICE PRINCIPAL AS FELLOW SERVANT.

The dual capacity doctrine of servant and vice principal may be invoked for the benefit of the vice principal as well as for the benefit of a fellow servant. A vice principal or foreman may have general charge and control of miners or employees, but where he is performing the same character of work as are the men under his control, and is injured by the negligence of another employee, the employer may be liable, if under the circumstances he is made liable for the negligence of a fellow servant. Where a vice principal or foreman was injured by being struck in the eye with pieces of rock or dirt caused by another employee and miner striking a bowlder with his hammer, the question of the negligence of the miner in striking the bowlder is one of fact to be determined by the jury.

Misenhelter v. Geronimo Lead & Zinc Co. (Missouri App.), 192 Southwestern, 147, p. 149.

METHODS OF OPERATING.**OPERATING UNDER LEASE.**

A coal-mining company, as owner with the right to operate a certain mine, entered into a contract with another as lessee and contractor, whereby the latter was to mine certain coal pursuant to an existing agreement. The lessee and contractor was to furnish the necessary equipment and material for the operation of a successful mine and to pay all costs and charges and wages in whatsoever nature involved in the mining operations, and to turn the mine back to the lessor in good condition. Under such a lease and agreement the lessee or contractor occupies the position of an independent contractor respecting the operation of the mine. Under such circumstances a person in the employ of the lessee and contractor and injured while working in the mine can not hold the lessor liable for the injuries under the workmen's compensation act of Kansas, where there is nothing to show that the work in which the plaintiff was engaged was under the direction, execution, or control of the lessor.

Maughliffe v. Price & Son (Kansas), 161 Pacific, 907.

MINING ON LANDS OF ANOTHER—LIABILITY.

The owner of certain coal lands sued the owner of adjoining lands operating a coal mine thereon for damages for taking coal. The plaintiff charged that the defendant by his superintendents, agents, miners, and employees entered upon the plaintiff's land adjoining and mined and removed therefrom a large quantity of the plaintiff's coal and converted the same to his own use. The evidence showed that the defendant leased to a third person the coal underlying his land, adjoining the land of the plaintiff; that in mining the coal the lessee of defendant trespassed upon the plaintiff's property and mined and removed a large quantity of coal belonging to the plaintiff; that the coal so mined by the lessee was carried to the surface through a coal mine belonging to the defendant and delivered at the mouth of the mine to the defendant in his own cars; that the defendant paid the royalty on the coal so mined by the lessee on the plaintiff's property, and that the defendant on demand for compensation for the coal taken made no disclaimer of responsibility and accountability, except that he had paid the royalty on the coal. The evidence also showed that during the period covering the trespass the lessee was superintendent of the defendant's mine. Under such circumstances and evidence the trial court could not disregard a verdict for the plaintiff and render judgment for the defendant.

Philson v. Wells (Pennsylvania), 100 Atlantic, 463.

DANGEROUS METHOD OF OPERATING MACHINERY.

An oil well drilling and pumping company is not liable in damages for injuries to an employee who adopted unnecessarily and admittedly dangerous methods of throwing in the clutch of a gas engine which was used in running the pump of an oil well, and by reason of which the injuries complained of were received.

Murphy v. Standard Oil Co., etc., 140 Louisiana, —; 73 Southern, 678, p. 679.

REMOVAL OF SUPPORT.

The right of an owner of the surface to support, whether by natural right or by agreement, imposes an absolute obligation upon the servient owner, regardless of any negligence on his part in making excavations or improvements. The right of subjacent support is a right of property passing with the soil, as otherwise the surface can not be enjoyed at all.

Audo v. Western Coal & Mining Co. (Kansas), 162 Pacific, 344, p. 346.

PURCHASE OF ENGINE—IMPLIED WARRANTY OF FITNESS.

A coal-mine operator hauled his coal cars and coal by horsepower from his mine to a shipping point, a distance of about 3,000 feet. He proposed changing the motive power to steam and entered into

negotiations with a manufacturer for the purchase of a locomotive for that purpose. The manufacturer visited the mine, examined the conditions, character of the track and the distance, and offered for a stated price to manufacture a locomotive engine suitable in size and capacity with the necessary appliances for properly and sufficiently hauling the operator's coal and coal cars. The offer was accepted, and the manufacturer constructed and delivered a locomotive engine. The engine was received and installed, and after repeated trials was found to be unfitted and wholly insufficient to do the work needed. A written order for the engine was made and signed by the mine operator that contained no express warranty that the engine should be suitable for the particular purpose, but under the circumstances the law creates an implied warranty under the rule that when a purchase is made for a particular purpose that is known to the seller, and especially where a manufacturer is informed as to the use which the buyer expects to make of the machine or implement and the work which he expects it to accomplish, there arises an implied warranty of fitness of the article, under proper management, to perform the work for which it was intended. The manufacturer knew the purpose for which the locomotive was being purchased and the use to which it was to be put, and had examined the premises and the track over which it was expected to run, and having manufactured the locomotive with this knowledge and in the absence of an express warranty, an occasion for the application of the doctrine of implied warranty is completely furnished, and a warranty is imposed by law to the effect that the locomotive manufactured for and furnished to the mine operator would, with proper management, fill the purpose for which it was bought; and failing to do this, the mine operator was not liable to the manufacturer for the price of the locomotive.

Glover Machine Works v. Cooke-Jellico Coal Co. (Kentucky), 191 Southwestern, 516, p. 518.

MINING LEASES.

LEASES GENERALLY—CONSTRUCTION.

LEASE AS AN INCUMBRANCE.

Under the statute of Colorado (R. S. 1908, sec. 865) prohibiting a mining corporation from encumbering its mining property without the consent of a majority of the stockholders, a lease is an encumbrance and is voidable at the election of the stockholders, if not approved or ratified by the holders of a majority of the shares of stock.

Elder v. Western Min. Co., 237 Fed., 966, p. 971.

WANT OF VALIDATION BY STOCKHOLDERS—EFFECT OF LACHES.

A mining corporation by its directors executed a lease of its mining property for a term of years without the approval of a majority of the stockholders as required by the statute of Colorado. The lease was extended from time to time for different periods without the knowledge of the stockholders. The last agreement for an extension was made three years and more before the extended term commenced. The stockholders did not learn of the extension until three years or more after the agreement and until nearly a year after the extended term began. A suit was commenced by the stockholders to avoid the lease within less than a year after they had knowledge of the unwarranted extension. The stockholders had the right to rely upon the faithful discharge by the trustees of their duties upon the legal presumption that they would not execute a lease, or an extension of a lease of the property, until approved by a majority vote of the shares of stock. No duty rested upon the stockholders to watch over or search out acts of their trustees, and neither trustees nor the lessee, who knew that the lease lacked the necessary statutory approval of the stockholders, can be permitted to defeat the suit of the stockholders to avoid the lease because the suit was not brought sooner. The delay, under the circumstances, is not sufficient laches, acquiescence, or ratification on the part of the stockholders to estop them from maintaining a suit to avoid the lease.

Elder v. Western Mining Co., 237 Fed., 966, p. 975.

CONSTRUCTION—CONFLICT BETWEEN PRINTED AND WRITTEN PARTS.

An oil and gas lease was executed on a printed or typewritten blank and words and parts were written in to make it complete. In certain parts of the lease there was a conflict between the printed

words of the blank form and the matter written in. In such case where there is reasonable doubt as to the sense and meaning of the entire lease, the words in writing will control the construction of the contract. The reason for this is that the written words are the immediate language and terms selected by the parties themselves, while the printed form was intended for general use without reference to particular objects.

Producers Oil Co. v. Snyder (Texas Civil App.), 190 Southwestern, 514, p. 515.

ROYALTIES FROM OPEN MINES.

Under a will providing for the operation of mines and for the exploration of other mineral lands and directing the method of the payment of royalties, no distinction can be made between open and unopened mines so far as concerns the royalties to be received under the terms of the will. The testator's purpose was to have all the leased mines opened and worked and leases were made by the testator under which ground rent was to be paid whether ore was mined or not, but if ore was mined, then the payment was to be in royalties, and the payment of royalties during any year was to discharge the ground rent provided for pro tanto for any such year.

Pettit's Estate, In re (Minn.), 161 Northwestern, 158, p. 162.

IMPLIED OBLIGATION TO DEVELOP.

Under the laws of Kentucky no one will be permitted to acquire and hold leases without development where the payment of a royalty upon the ore produced is provided for in the terms of the lease.

United Mining Co. v. Morton (Kentucky), 192 Southwestern, 79, p. 84.

CERTAINTY AS TO TIME.

A city by a lease with a landowner acquired the natural gas rights to a tract of land and agreed to pay therefor an annual rent of \$200. The lease provided that it should remain in force for the same length of time as another lease covering gas rights to an adjoining tract upon which the city was then operating natural gas wells. This reference to another lease sufficiently fixed the duration of the lease in controversy and it is not subject to the objection that it is indefinite and uncertain and therefore void.

Butler v. City of Iola (Kansas), 163 Pacific, 652.

ORAL SURRENDER.

The owner of a mining lease can not assign his rights to a stranger except by a writing, but a lessee in a mining lease may orally surrender the lease to the lessor.

United Mining Co. v. Morton (Kentucky), 192 Southwestern, 79, p. 84.

MACHINERY—REMOVAL.

Under a lease of mining property the parties may by their contract stipulate what machinery and fixtures may be removed and when they have done so such stipulations are controlling. Mining machinery, apparatus and appurtenances placed upon mining property by a lessee are not regarded as fixtures that pass with the soil and leased appurtenances, but are regarded as the personal property of the lessee that may be removed by him in the absence of an express stipulation in the lease to the contrary.

Shaleen v. Central Coal & Coke Co. (Arkansas), 192 Southwestern, 225, p. 227.

FIXTURES—RIGHT TO REMOVE TIPPLE.

A mining lease provided that at its expiration the lessee could remove all machinery, pit cars, mine rails, and pipe that had been placed on the mining premises, if the lessee had performed the terms and covenants of the lease, but all other property on the leased premises at the expiration of the lease shall belong to and revert to the lessor. On the expiration of the lease, the lessee removed the mining machinery and also the tippie erected by him on the leased premises. In an action by the lessor to recover possession of the tippie, it was proper for the court to submit to the jury the question whether the tippie was a part of the mining equipment and removable as such under the lease.

Shaleen v. Central Coal & Coke Co. (Arkansas), 192 Southwestern, 225, p. 227.

RIGHT TO REMOVE MACHINERY AND FIXTURES.

Where a mining lease gave the lessee the right at its expiration to remove from the leased premises all machinery, pit cars, mine rails, etc., the right to remove the machinery stipulated does not expire at the instant of the expiration of the lease, but the lessee where actively engaged in removing the machinery has a reasonable time to continue the removal after the lease actually expired.

Shaleen v. Central Coal & Coke Co. (Arkansas), 192 Southwestern, 225, p. 227.

OIL AND GAS LEASES.**CONSTRUCTION—SEPARATE LEASES.**

An oil and gas lease embraced 19 quarter sections of land and provided that the drilling of a well or wells should stop rent only upon the quarter section or sections upon which such well or wells are located. According to the terms of the lease, it must be construed as a separate lease of each quarter section, requiring the sinking of a well on each quarter section in order to avoid forfeiture.

Producers Oil Co. v. Snyder (Texas Civil App.), 190 Southwestern, 514 p. 516.

CONSTRUCTION—ROYALTIES.

A gas lease provided a graduated scale of royalties based on a minute's pressure and the scale contained a minimum pressure but was silent as to the use of gas or payment therefor while the pressure was below the minimum. The lease did not limit the right of the lessee to take gas only so long as the pressure should exceed the minimum of 15 pounds; but there is no provision for the payment of royalty when the pressure falls below the minimum. Under the lease and the standard for the payment of royalties, the lessor is not entitled to royalties for gas produced below the minimum pressure, though it was used or marketed by the lessee.

Adderman v. Manufacturers Light & Heat Co. (Pennsylvania), 100 Atlantic, 444, p. 446.

An oil and gas lease bound the lessee to deliver to the credit of the lessor in a pipe line one-eighth of the oil produced when the average daily production of the wells did not exceed 10 barrels; one-sixth when the production exceeded 10 barrels, but did not exceed 40 barrels, and one-fourth when the production exceeded 40 barrels. There is no evidence of mutual mistake in the execution of the lease in not basing the royalties upon the average daily production of a single well as a unit, such as will justify a court of equity in reforming the instrument.

Siluran Oil Co. v. Neal (Illinois), 115 Northeastern, 114, p. 115.

CONSTRUCTION—OBLIGATION.

A mining lease executed by several lessors or grantors by which they disposed of the mineral rights on their several tracts of land for a gross sum without stating the amount paid to each grantor or designating the area of land belonging to each, creates a general obligation on the part of such lessors, because it is important to affirm that the lessee would have paid a proportionate consideration for the lease of the mineral rights on only a portion of the land. A stipulation in such a lease to the effect that operations for the drilling of a well for oil or gas shall be commenced by the lessee within one year can not be construed to mean that the operations for the drilling of a well shall be commenced on each separate tract of land belonging to the different lessors.

Nabors v. Producers Oil Co., 140 Louisiana, —; 74 Southern, 527, p. 532.

CONSTRUCTION—OBLIGATION TO DRILL.

In a lease of land for the production of oil and gas in which the lessee expressly obligated himself to begin the drilling of a well within one year or forfeit the lease, there is no implied obligation on his part to drill as many wells as may be reasonably necessary to secure the

oil or gas for the common advantage of the lessor and lessee within the year, where oil or gas has not been found in paying quantities.

Nabors v. Producers Oil Co., 140 Louisiana —, 74 Southern, 527, p. 532.

CONSTRUCTION OF LEASE—INTEREST OF LESSEE.

An ordinary oil and gas lease giving a lessee the right to enter upon lands and drill for oil and gas and other minerals, does not vest in the lessee the title to the oil and gas in the land and is not a grant of any estate therein, but is only a grant of the right to prospect for oil, and gas, and no title vests until these minerals are reduced to possession by extracting them from the earth.

Kelley v. Harris (Oklahoma), 162 Pacific, 218, p. 220.

PRACTICAL CONSTRUCTION.

Where the construction of an oil and gas lease or other contract is in doubt the acts of the parties thereunder may frequently be resorted to for the purpose of ascertaining how they construed the instrument. Under this rule the court will adopt the construction placed upon an oil and gas lease by the lessee and that he abandoned the same when he was no longer entitled to drill upon the premises by virtue of the lease.

Kelley v. Harris (Oklahoma), 162 Pacific, 218, p. 221.

LEASE CONSTRUED AGAINST LESSEE.

Where the language of an oil and gas lease was as much that of the lessee as that of the lessor the lease will be construed most strongly against the lessee in order to provoke development and prevent delay and unproductiveness looking to all parts of the instrument in the light of the facts in connection with the operation.

Paraffin Oil Co. v. Cruce (Oklahoma), 162 Pacific, 716, p. 718.

NATURE OF LEASE—INTEREST OF LESSEE.

A lease of land for oil and gas obligated the lessee to pay a stipulated royalty for all the oil or gas that might be produced, and provided that under penalty of forfeiture the lessee should commence the drilling of one well within one year of the execution of the lease or pay a stipulated consideration each year for an extension of the time not exceeding a period of three years in all, and provided also that if the lessee discovered oil and gas within the time limit or within the extension of the time limit as provided the lease shall remain in full force and effect for 20 years from the discovery of oil or gas and as much longer as minerals are produced in paying quantities. The lessee in addition to the stipulations in the lease paid an adequate cash consideration. Such a lease is a conveyance of a real right and can not be construed as a sale of a mere hope of producing oil or gas

within one year of its execution; and where the lessee complied with his obligation by commencing the drilling of a well within the year from the execution of the lease and continued to prosecute the work with due diligence the lessor is not entitled to a cancellation of the lease at the expiration of the year on the ground that he fully discharged his obligation by permitting the lessee to attempt to realize his hope within the year.

Nabors v. Producers Oil Co., 140 Louisiana, —; 74 Southern, 527, p. 534.

INTERESTS AND RIGHTS OF LESSEE.

In the absence of a specific covenant in an oil and gas lease making the lessee liable for damages to growing crops and their surface rights the lessee is not liable for such damages as are necessarily incident to the operations authorized by the lease. Such a lease carries within its implications, if not within its expression, such rights to the surface as may be necessarily incident to the performance of the objects of the contract, yet these implications go no further and the lessee must protect the surface of the ground in so far as such incident necessity does not exist and is liable to the lessor for any damages to the surface resulting from acts not within the implications of the lease.

Pulaski Oil Co. v. Conner (Oklahoma), 162 Pacific, 464, p. 466.

PRIORITY OF LEASES—NOTICE OF PRIOR LEASE.

The question as to whether or not an original oil and gas lease was not properly acknowledged and was not entitled to record, and was not, therefore, when recorded, constructive notice, is immaterial in a controversy between the original lessee and junior lessees where it is shown that the junior lessees had actual knowledge of the original lease and that such original lessee had paid the rentals as provided in his lease, and where the second lessee orally agreed to protect the lessors against any litigation or damages by reason of the original lease.

Mexico-Wyoming Petroleum Co. v. Valentine, 237 Fed., 539, p. 545.

PRIORITY OF LEASE—NOTICE—PROOF OF DEVELOPMENT.

In an action by the holder of an oil and gas lease to enjoin operations by a subsequent lessee of the same property proof on the part of such junior lessee to the effect that by their labors they had increased the value of the property from \$40 an acre to about \$5,000 an acre is improper and inadmissible where it appears that such subsequent lessees took their lease and performed the labor with full notice and knowledge of the first lease, and that two wells, of the wells claimed to enhance the value of the property, were drilled after the commencement of the suit by the holder of the first lease.

Mexico-Wyoming Petroleum Co. v. Valentine, 237 Fed., 539, p. 542.

FAILURE TO DEVELOP.

A delay in the development of an oil and gas lease for a period of 16 months on the part of a first lessee is far within the statute of limitations in an action at law, and the burden of showing that the enforcement of a prior lease as against subsequent lessees with notice would be inequitable is upon such subsequent lessees. Mere lapse of time does not ordinarily constitute laches, but the situation of the parties must have so changed as to render the enforcement of the first lease inequitable as against the subsequent lessees.

Mexico-Wyoming Petroleum Co. v. Valentine, 237 Fed., 539, p. 543.

IMPLIED OBLIGATION TO DEVELOP.

There is an implied condition in every lease of land for the production of oil that when the existence of oil in paying quantities is made apparent the lessee shall put down as many wells as may be reasonably necessary to secure the oil for the common advantage of both lessor and lessee, but this doctrine has no application to a lease where there has been no discovery of oil or gas within the delay period.

Nabors v. Producers Oil Co., 140 Louisiana, —; 74 Southern, 527, p. 533.

DILIGENCE IN OPERATIONS—DETERMINATION.

Where the object of the operations contemplated by an oil and gas lease is to obtain a benefit or profit for both lessor and lessee neither is, in the absence of a stipulation to that effect, the arbiter of the extent to which or the diligence with which the operations shall proceed, but both are bound by the standard of what under the circumstances would be reasonably expected of operators of ordinary prudence, having regard to the interest of both.

Paraffin Oil Co. v. Cruce (Oklahoma), 162 Pacific, 716, p. 721.

POWER OF LESSEE TO TERMINATE.

A lessee in an oil and gas lease can not set up his own default in order to terminate the lease or escape liability under its provisions. If he fails to perform the covenants of the lease it is with the lessor to declare a forfeiture. Where such a lease provides that if the premises should not be operated the lease should be void the word "void" means "voidable" at the election of the lessor and he must do some act evincing an intention to avoid the lease before it can be considered void or terminated. Such provisions are for the benefit of the lessor and he has an option to discontinue the lease on default of the lessee or affirm the continuance of the contract. If such a lease provides that the lessee's failure to complete a well within a stated period or any default thereof to pay a certain yearly rental should render the lease null and void and all rights and claims should

therefrom cease, still the lessee by his own default can not relieve himself from the liability already incurred.

Lavery v. Mid-Continent Oil & Development Co. (Oklahoma), 162 Pacific, 737, p. 739.

POWER OF LESSEE TO CANCEL—LIABILITY.

A landowner leased to a city the gas rights to a tract of land at an annual rental of \$200. The lease provided that it should remain in force for the same length of time as another lease held by the city giving it gas rights to an adjoining tract of land and upon which the city was then operating gas wells. After paying the annual rental for seven years, and while still operating gas wells on the adjoining lands under the lease referred to, the city has no power to cancel the lease in question by executing and recording a release thereof and refusing to make further payments of rental on the ground that the gas rights obtained by the lease are of no value. The city by the lease in question obtained more than a mere license to explore for oil and gas and to pay upon certain contingencies a royalty, but by the lease in question the city obtained the gas rights of a tract of land and agreed to pay an annual rental therefor so long as the lease remained in force. The inducement for obtaining the lease in question was to protect the rights of the city under the lease on the adjoining land is apparent from the fact that the city sunk no wells on the land covered by the lease.

Butler v. City of Iola (Kansas), 163 Pacific, 652, p. 653.

AGREEMENT TO ASSIGN LEASE—CANCELLATION.

Two persons held separate interests in leases for different tracts of oil lands. One lessee agreed to assign to the other an undivided one-half interest in leases held by him and the other agreed to assign his interest in leases held by him. The latter agreed also that any profits or benefits that he might derive from a contract between himself and a certain oil company should be equally divided between the two persons. It was also agreed that the latter should pay to the former a salary of \$250 per month from the date of the contract and that the first named person should operate all of the property under the supervision of the second. It was agreed that the second party might cancel the contract at any time. In an action by the second party to compel the first party to the agreement to execute the conveyance of the leases as contemplated in the agreement, parol evidence is inadmissible to show that a subsequent written agreement entered into between the parties was intended to cancel the former, where the subsequent written agreement itself presented conclusive evidence that it was not intended as a complete extinguishment of the former agreement.

Piper v. Kellerman (California), 162 Pacific, 423, p. 425.

FORFEITURE—FAILURE TO DEVELOP.

An oil and gas lease provided that it should remain in force for the term of one year from its date and as long thereafter as oil or gas is produced from the premises by the lessee; and providing that "if said territory proves to be productive, then the party of the second part to complete this contract shall drill as many as eight wells on said premises and said wells shall be drilled with due diligence and dispatch having in view the interest of both parties hereto, and so to produce all the oil or gas that may be reasonably produced from said premises." The lessee proceeded immediately and within 30 days drilled a productive well upon the leased premises. It then became his duty to proceed immediately to drill the eight wells as contemplated by the lease. The word "if" as used in the quoted clause means "when" and the word "then" used in the quoted clause, is an adverb of time and means "at the time," that is, at the time the territory proved productive by the drilling of the test well it was then the duty of the lessee to drill as many as eight wells upon the leased premises and within a year from the date of the lease as a condition precedent to the extension of the lease beyond the term of one year. Whether such wells were by the lessee drilled with due diligence and dispatch, having in view the interest of both parties to the lease and so as to produce all the oil and gas that may be reasonably produced from the premises as required by the lease, is a question of fact to be determined in connection with all the circumstances attending the operations. The fact that a test well was profitable and that there was at its completion a profitable market make the failure of the lessee to drill as many as eight wells with due diligence sufficient ground for the forfeiture of the lease on the part of the lessor.

Paraffin Oil Co. v. Cruce (Oklahoma), 162 Pacific, 716, p. 717.

Lavery v. Mid-Continent Oil and Development Co. (Oklahoma), 162 Pacific, 737.

FAILURE TO DEVELOP—RIGHT TO CANCELLATION.

When the lessee of land for the production of oil or gas has paid an adequate cash consideration and has complied with the only obligation expressly required of him during the first year after the execution of the lease, by the drilling of the one test well required and has not found oil or gas in paying quantities, the lessor can not compel a cancellation of the lease on the ground that the lessee failed to perform an implied obligation to drill more than one well for the common advantage of the lessor and lessee.

Nabors v. Producers Oil Co., 140 Louisiana —; 74 Southern, 527, p. 533.

FORFEITURE—FAILURE TO EXECUTE RELEASE.

In an action by a landowner against a lessee who had forfeited his lease for damages on the ground of his failure and refusal to execute a release and remove the cloud from the title, the landowner as complainant must prove that he had a valid contract with a prospective purchaser or lessee enforceable according to its terms, had the release by the defendant been executed, or that such prospective purchaser would have completed his contract of purchase or lease regardless of its enforceability, had such release been executed.

Rogers v. Milliken Oil Co. (Oklahoma), 161 Pacific, 799, p. 800.

OPTION TO FORFEIT.

By the terms of an oil and gas lease the lessee, an oil company, for a valuable consideration specifically undertook to commence and with diligence drill a well on the premises into a designated sand. The lease contained a clause providing that a failure to commence and complete said well should work a forfeiture and render the lease null and void. This forfeiture provision was for the benefit of the owner of the leasehold interest and gave him the option to declare a forfeiture upon the failure of the oil company to discharge its obligation to drill. The oil company could not by virtue of the forfeiture clause and without the consent of the owner terminate the contract by its own default and thereby escape liability for resultant damages.

Lavery v. Mid-Continent Oil & Development Co. (Oklahoma), 162 Pacific, 737, p. 739.

CONSTRUCTION—OPERATION AND DEVELOPMENT.

Leases executed for the purpose of exploring and operating for oil and gas frequently contain provisions designated to compel the prompt commencement of operations and the diligent prosecution of the development of the land. In the construction of such leases the courts endeavor to discover and enforce the intention of the parties as gathered from the language used in the instrument as a whole. When the terms of such a lease will permit, it will be so construed as to promote development and prevent delay and unproductiveness.

Paraffin Oil Co. v. Cruce (Oklahoma), 162 Pacific, 716, p. 718.

ABANDONMENT—REMOVAL OF MACHINERY.

The lessee under an oil lease who drills a well which proves unprofitable, may abandon the lease and land and may within a reasonable time exercise the right conferred by the lease to remove the machinery, fixtures, and improvements made upon the land and may remove pipe placed in the ground by him in his mining operations. Eight months from such abandonment is a reasonable time within which to remove the machinery and fixtures and pipe.

Standard Oil Co. of Louisiana v. Barlow, 141 Louisiana —; 74 Southern, 627.

ACTION FOR ROYALTIES.

In an action by a lessor against the lessee of an oil and gas lease for royalties it is not improper for the court to instruct the jury as to the amount the lessor claims as due and owing for gas furnished and marketed by the defendant under the terms of the agreement sued upon, as such an instruction is not a statement of any fact but is only a statement of the plaintiff's claim.

Laurel Oil & Gas Co. v. Anthony (Oklahoma), 162 Pacific, 203, p. 204.

TEST—METHOD OF MEASURING GAS.

An oil and gas lease provided that the lessee should pay to the lessor for any wells producing gas in sufficient quantities to justify the lessee in marketing the same in its pipe line, a semiannual sum to be ascertained by gaging the wells in the casing in which they were completed every six months, and if a well should show for the first minute a pressure of 200 pounds or more the rental should be \$250 for the next six months; if the wells should show a pressure from 150 to 200 pounds, the rental to be \$200, and on down according to a graduated scale to a minimum pressure of 15 to 25 pounds and a rental of \$25. The lease did not determine when or designate the pipe or casing in which the well should be completed and according to the evidence the words used have no defined trade meaning. Under the peculiar wording and the operation of the lease, the question as to whether a well was completed in a 3-inch casing or in a 6½-inch casing is one of fact to be determined by the jury.

Adderman v. Manufacturers Light & Heat Co. (Pennsylvania), 100 Atlantic, 444, p. 445.

MINING PROPERTIES.

TAXATION.

CONSTRUCTION AND APPLICATION OF STATUTE.

The Legislature of Nevada in 1905 (Statutes 1905, p. 81), passed an act authorizing assessors to assess patented mines. The statute points for its authority directly and specifically to the constitutional amendment adopted at the general election in November, 1902, and takes its constitutional authority and its operative vitality directly from the constitutional amendment that provides for the assessment of patented mines at a flat valuation of \$10 per acre.

Wren v. Dickson (Nevada), 161 Pacific, 722, p. 725.

CONSTITUTIONAL PROVISIONS—SELF-EXECUTING.

The constitution of Nevada was amended at the general election in November, 1902. The amendment provided that patented mining claims should be assessed at a valuation of \$10 per acre. The legislature of 1905 (Statutes 1905, p. 81) passed an act, pursuant to the mandate of the amendment authorizing assessors to assess patented mines at a flat valuation of \$10 per acre. The constitution was again amended at the general election in 1906, by which it was provided that the proceeds alone of unpatented mining claims should be assessed and taxed, and when patented each patented mine should be assessed at not less than \$500, except when \$100 in labor has been actually performed on such patented mine during the year in addition to the tax upon the net proceeds. This provision of the amendment of 1906 is self-executing and in effect nullified the amendment of 1902 as to the assessment of patented mines at a flat valuation and rendered nugatory and of no effect the statute of 1905, authorizing the assessment of patented mining claims at a flat valuation of \$10 per acre. After the adoption of the amendment of 1906 an assessment of a patented mining claim could not be made under the act of 1905 that was based on the amendment of 1902, and any attempted assessment under the act of 1905 made subsequent in time to the adoption of the amendment of 1906 was invalid.

Wren v. Dickson (Nevada), 161 Pacific, 722, p. 725.

TAXATION WHERE ANNUAL ASSESSMENT LABOR HAS BEEN PERFORMED.

The amendment of the constitution of Nevada at the general election of 1906 (Revised Laws, sec. 352) provided that each patented mine shall be assessed at not less than \$500, except when \$100 in labor has been actually performed. A mine or mining claim upon

which \$100 worth or more of labor has been expended during any one year and prior to the time of assessment is exempt from taxation except on the proceeds thereof.

Wren v. Dickson (Nevada), 161 Pacific, 722, p. 725.

METHODS OF ASSESSMENT.

The constitutional amendment of 1906 (Nevada Revised Laws, sec. 352) contains two distinct negatives as to the assessment of mining claims: (1) A patented mine can not be assessed at less than \$500, if \$100 worth of work has not been performed; (2) A patented mine on which \$100 worth of labor has been performed can not be assessed at either more or less than \$500, but is exempt from taxation except on the proceeds thereof. Where the annual labor has been performed assessment for any sum is prohibited. Where the annual labor has not been performed, the assessment for a sum less than \$500 is prohibited. These prohibitory provisions of the constitutional amendment are self-executing.

Wren v. Dickson (Nevada), 161 Pacific, 722, p. 728.

TAX ON OIL PRODUCTION—FEDERAL AGENCY.

The statute of Oklahoma (Session Laws 1916, p. 102) imposes a tax equal to 3 per cent of the gross value of the production of petroleum or other crude or mineral oil and natural gas less the royalty interest. It also provides that the payment of such tax shall be in full and in lieu of all taxes by the State, county, city, township, or other municipalities upon any property rights attached to or inherent in the right to such minerals, upon leases for the mining for petroleum or other crude oil or other mineral oil, or for natural gas, upon the mining rights and privileges belonging to or appertaining to land, upon the machinery and equipment used in or around any well producing petroleum or other crude or mineral oil or natural gas and actually used in the operation of a well. This statute is not subject to the objection that because the business was carried on and the production brought about through the instrumentality of a Federal agency, the State is without power to impose a tax. The tax provided for by this act is not upon the agency or the means employed by the producer, but is upon the production of oil and gas, and is therefore a "tax on property" as such and is valid without regard to the agency employed in its production. The act does not impose a tax upon the operations of an oil company, or upon its right to engage or continue in business, but upon the commodity which it produces and the tax is made in full and in lieu of certain of its other property and as a substitute therefor. The tax is payable upon the gross value of the oil and gas produced without regard to the value of the other enumerated property of the producer in lieu of which the tax is imposed.

Large Oil Co. v. Howard (Oklahoma), 163 Pacific, 537, p. 539.

TAXATION—CONSTRUCTION OF STATUTE.

The statute of Oklahoma (Session Laws 1916, p. 102), imposes a tax equal to 3 per cent of the gross value of the production of petroleum or other crude or mineral oil and natural gas less the royalty interest. It also provides that the payment of such tax shall be in full and in lieu of all taxes by the State, county, city, township, or other municipalities upon any property rights attached to or inherent in the right to such minerals, upon leases for the mining for petroleum or other crude oil or other mineral oil, or for natural gas, upon the mining rights and privileges belonging to or appertaining to land, upon the machinery and equipment used in or around any well producing petroleum or other crude or mineral oil or natural gas and actually used in the operation of a well. This statute does not purport to tax the leases or the mining rights or privileges or any investments therein. The tax is laid upon the oil and gas produced without regard to the value of the property or mining rights, leases, privileges, machinery, appliances, and equipment used in or around any producing well. The lease through which the rights or privileges are secured furnishes only the evidence of title or ownership and not a subject of taxation.

Large Oil Co. v. Howard (Oklahoma), 163 Pacific, 537, p. 543.

TAX ON OIL AND GAS—CONSTITUTIONALITY OF STATUTE.

The tax imposed by the act of the Oklahoma Legislature of May 14, 1916 (Session Laws 1916, p. 102), is not, because of its amount, repugnant to section 9, article 10, of the constitution, which provides that the State levy on an ad valorem basis shall not exceed in any one year $3\frac{1}{2}$ mills on the dollar. This act imposes a special tax on oil and gas in lieu of and as a substitute for the general ad valorem tax on certain property of the producer and is levied pursuant to sections 13 and 22, article 10, of the constitution, which authorizes the State to select its subjects of taxation, classify and value property so selected by means and methods different from that commonly employed in the assessment, levy and collection of taxes by municipal authorities generally.

Large Oil Co. v. Howard (Oklahoma), 163 Pacific, 537, p. 547.

TAX ON OIL AND GAS—EFFECT ON INDIAN TREATY RIGHTS.

The act of the legislature of Oklahoma of May 14, 1916 (Session Laws 1916, p. 102), imposing a tax of 3 per cent on the gross value of the production of oil and gas can not deprive an oil company of its power to serve the Indians or the General Government acting in their behalf. The act does not impair the treaty rights of the Osage Indians for whom the federal Government lawfully undertakes to act and it is too remote and indirect to be regarded as an

interference with the Federal agency or a direct burden upon its free exercise.

Large Oil Co. v. Howard (Oklahoma), 163 Pacific, 537, p. 547.

CONSTRUCTION AND APPLICATION OF STATUTE.

The act of the Oklahoma Legislature of May 14, 1916 (Session Laws 1916, p. 102), imposes a tax of 3 per cent on the gross value of the production of petroleum or crude oil and of natural gas, less the royalty interest. This statute must not be confounded with the act of May 26, 1908 (Session Laws 1907-8, p. 640) that provides a gross revenue tax which shall be in addition to the taxes levied and collected upon an ad valorem basis upon the property and assets of an oil company equal to a per centum of its gross receipts as therein provided. Then, while the present act provides that the tax is in full and in lieu of all taxes, the act of 1908 by express terms makes the tax "in addition to the taxes levied and collected upon an ad valorem basis."

Large Oil Co. v. Howard (Oklahoma), 163 Pacific, 537, p. 546.

INVALID TAX SALE—RECOVERY—STATUTE OF LIMITATIONS.

The amendment of the constitution of Nevada at the general election of 1906 (Nevada Revised Laws, sec. 352), prescribing the method of the assessment and taxation of patented mines and mining claims completely nullified the amendment of 1902 and the act of the legislature of 1905 (Nevada Statutes 1905, p. 81), authorizing the assessment of patented mining claims at a flat valuation of \$10 per acre. Any assessment of a patented mining claim made under the act of 1905, after the adoption of the amendment of 1906, was void, and any subsequent sale for taxes on such an assessment was wholly void and of no effect whatever. Under the statutes of Nevada, an action to recover a mining claim must be brought within two years, and, though a tax sale should be wholly invalid, the original owner must commence his action to recover the claim within the statutory period unless the lawful owner was under some disability, and in such case the period of disability can not be a part of the time limit for the commencement of the action; but in such case an action may be commenced within two years after the removal of the disability.

Wren v. Dickson (Nevada), 161 Pacific, 722, p. 731.

TRESPASS.

PROOF OF DAMAGES—LIMITATION AS TO TIME.

In an action for damages for trespass to a mine and mining property it is not proper to prove any damages or depreciation in the mining property prior to the time the complainant acquired title. In esti-

inating or arriving at the damages the jury can not base its estimate on the theory adopted and given by witnesses that covering a period of 10 years, 40 per cent of the damages and depreciation was to be charged to the first four years and 60 per cent to be charged to the remaining 6 years, where the evidence shows that the defendant was in possession for the six years only.

Llewellyn v. Sunnyside Coal Co. (Pennsylvania), 99 Atlantic, 869, p. 870.

RENTAL VALUES—CROSS-EXAMINATION OF WITNESS.

In an action of trespass to recover damages for injuries to a coal mine and mining property and for the rental value of land in connection with the mining property, a witness who gives his opinion as to the rental values based on his knowledge of the sales of such property may be asked on cross-examination as to rentals paid for the properties referred to as a test of his good faith and credibility.

Llewellyn v. Sunnyside Coal Co. (Pennsylvania), 99 Atlantic, 869, p. 871.

REMOVAL OF ORE—FRAUD—STATUTE OF LIMITATIONS.

The Golden Eagle Mining Co. and the Emperor-Quilp Co. were the owners of adjoining mining claims. Between September 1910 and August 1912, the Emperor-Quilp Co. exposed and developed their vein of ore to the point of intersection with the boundary line between its claim and that of the Golden Eagle Mining Co. at the depth of between 300 and 500 feet below the surface of the ground. The Emperor-Quilp Co. by its officers and employees knowingly and wrongfully entered upon and into the claim of the Golden Eagle Mining Co. and extracted and removed ore therefrom of the value of several thousands of dollars. At that time there were no underground workings of any character upon the claim of the Golden Eagle Mining Co. and it did not discover, nor have any means of discovering such wrongful removal of the ore until December, 1912. In December, 1915, more than three years after the wrongful removal of the ore, but less than three years after its discovery, the Golden Eagle Mining Co. commenced an action against the Emperor-Quilp Co. to recover the value of the ore so wrongfully removed. Under the statute of Washington an action for waste or trespass upon real property must be commenced within three years after the cause of action has accrued. But in actions for relief upon the ground of fraud the cause of action is not deemed to accrue until the discovery thereof by the aggrieved party. In some States, notably Pennsylvania and California the courts hold that the secret taking of ore from beneath the surface of the ground is in effect a fraud upon the rights of the owner of the property trespassed upon and such as gives him a cause of action for relief upon the ground of

fraud thereby changing the rule as to the statute of limitations. The Supreme Court of Washington made no distinction between the wrongful taking of ore between 300 and 500 feet below the surface from an unworked mining claim and that of an ordinary fraud practiced in the open where detection might follow without delay, and accordingly held that the action was barred in the three years from the time the trespass was committed.

Golden Eagle Mining Co. v. Imperator-Quilp Co. (Washington), 161 Pacific, 848.

See *Lewey v. Fricke Coal Co.*, 166 Pa. 536, 31 Atlantic 261, 45 Am. St. 684, 28 L. R. A. 283.

Lighthner Min. Co. v. Lane, 161 California 689, 120 Pacific, 771.

Lindley On Mines (3d ed.) paragraph 867.

Langley v. Devlin (Washington), 163 Pacific, 395.

LIENS.

FORECLOSURE OF MORTGAGE—SUPERIOR TITLE.

Ordinarily an adverse title to mining property paramount to a mortgage can not be litigated in a proceeding to foreclose the mortgage; but the litigation of the title in such proceeding is not without jurisdiction though it may be error if it be over the objection of the owner of the paramount title. But if the complaint in the foreclosure proceedings alleges that the defendant claims an interest or a lien which if valid is subsequent to the mortgage, the defendant is bound to assert his permanent title, or his interest or lien and on failure to do so he is bound by a decree unless appealed from.

Dickerman Inv. Co. v. Oliver Iron Min. Co. (Minnesota), 160 Northwestern, 776, p. 778.

QUARRY OPERATIONS.

FELLOW SERVANTS.

An employee in a stone quarry was pushing an empty car upon the main track from the crusher to the place in the quarry at which it was loaded. Shortly before he reached the part of the main track intersected by the spur two other employees of the company in charge of a loaded car that was standing chocked on the spur, carelessly and negligently removed the chock thereby permitting the loaded car to run down the spur to the main track where it collided with the empty car and injured the employee pushing the empty car. The employees operating these two separate cars were not fellow servants within the rule that would defeat the injured employee's right of recovery. They were fellow servants only in the sense that all were employed by the same master and engaged in similar services but the operators of the empty car had no control or direction over the operators of the loaded car and the two employees pushing the empty car were not so associated with the two persons operating the loaded car as that they could, under the circumstances, and by the exercise of ordinary care and attention, protect themselves from the negligent acts of the operators of the loaded car.

Cummins v. Sparks Coal Co. (Kentucky), 191 Southwestern, 515.

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BULLETIN 118. Abstracts of current decisions on mines and mining, reported from October to December, 1915, by J. W. Thompson. 1916. 74 pp.

BULLETIN 126. Abstracts of current decisions on mines and mining, reported from January to April, 1916, by J. W. Thompson. 1916. 90 pp.

BULLETIN 147. Abstracts of current decisions on mines and mining, reported from September to December, 1916, by J. W. Thompson. 1917. 84 pp.

TECHNICAL PAPER 138. Suggested safety rules for installing and using electrical equipment in bituminous coal mines, by H. H. Clark and C. M. Means. 1916. 36 pp.

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BULLETIN 143. Abstracts of current decisions on mines and mining, reported from May to August, 1916, by J. W. Thompson. 1917. 72 pp. 10 cents.

TECHNICAL PAPER 53. Proposed regulations for the drilling of oil and gas wells, with comments thereon, by O. P. Hood and A. G. Heggem. 1913. 28 pp., 2 figs. 5 cents.







Bulletin 153

DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

**THE MINING INDUSTRY IN THE TERRITORY
OF ALASKA**

DURING THE CALENDAR YEAR 1916

BY

SUMNER S. SMITH

United States Mine Inspector



**WASHINGTON
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1917**

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THE MINING INDUSTRY IN THE TERRITORY OF ALASKA DURING THE CALENDAR YEAR 1916.

By SUMNER S. SMITH.

INTRODUCTION.

The year 1916 broke all previous records of mineral production in the Territory. As a consequence, there has been a noteworthy increase in the number of mines in active operation or under development, though the greater part of the increased output is the result of greater activity at mines already producing rather than of opening new mines.

Owing to available funds not being sufficient to meet the cost of travel in the interior of the Territory during the winter months, the Federal mine inspector was unable during the spring to visit the placer mines of the interior as has been his custom in the past. However, by working in close cooperation with the Territorial inspector, who left Juneau in June to take the first boat down the Yukon to Fairbanks after the spring break-up, it was possible to visit hurriedly all the larger mining centers of the Territory. The Territorial inspector covered the districts from Fairbanks to Nome and the Federal inspector covered those of southeastern and southwestern Alaska.

A number of applicants have begun mining coal on ground held under free-use 10-acre permits, and have furnished coal for local use. The larger leasing units in the Bering River and Matanuska coal fields have been advertised and applications for leases have been submitted. These applications came too late in the fall for active development to begin before the end of the year.

The Government railroad has been built from Anchorage to the center of the Matanuska coal field, and private interests have undertaken the building of a standard-gage road from a point known as Goose City on Controller Bay to Canyon Creek, in the Bering River field. Capital has become interested in the oil fields on Controller Bay, and plans have been made for more active exploration and production.

The work of the Federal mine inspector is still handicapped by the lack of funds for additional field assistants and of the office accommodations necessary for carrying on the work with the efficiency that its importance warrants.

ACKNOWLEDGMENTS.

The preparation of this report has involved the compilation of data from many sources, and the Federal inspector gratefully acknowledges the assistance of all those who have furnished information. He extends thanks to members of the United States Geological Survey, the Alaskan Engineering Commission, the General Land Office, the Department of Justice, the officials of many mining and steamship companies, and individual miners, prospectors, and engineers. Particular credit is due Roy A. Dye, clerk to the Federal mine inspector; William Maloney, Territorial mine inspector; George R. Goshaw, of Chitina; C. H. Kraemer, George F. White, M. J. Callaghan, and M. B. Vaughn, of Valdez; R. G. Doherty, of Anchorage; and W. H. Dugdell, of Katalla.

NEW DISTRICTS.

In only one district were the discoveries of placer gold rich enough to attract unusual attention, though the limits of the known profitable areas in many of the older fields were considerably extended.

In the fall of 1915 rich discoveries were reported in the Tolstoi district, in the Innoko Basin, and during 1916 a small stampede carried a number of prospectors to that locality. So far the discoveries have not been large enough to warrant an opinion on the possible future of the district, although excellent pay was reported on Boob Creek.

The Yukon Gold Co. installed a dredge on Greenstone Creek, in the Ruby district. Although pay had been found on many of the creeks of that locality the erection of this dredge extended the limits of the district considerably and will add materially to the life of the community.

The Nelchina district ceased to exist, so far as mining was concerned, and the number of operators in the Chisana dwindled to half its former size.

The Marshall district, on the lower Yukon, showed considerable improvement, as did the Tolovana, about 50 miles northeast of Fairbanks. The best pay found in the latter district is on Liven-good Creek. The short, rainy season held back work somewhat, as the water for sluicing was very limited. The post office in this district has been named Brooks.

MINERAL PRODUCTION.

The Federal inspector has not attempted to present a comprehensive collection of data on the mineral output of the Territory as it would duplicate the excellent work of the United States Geological Survey. Hence he has taken the following figures on mineral pro-

duction from the advance statement of Alfred H. Brooks, of the Survey:

The value of the total mineral production of Alaska in 1916 is estimated to have been \$50,900,000; that of 1915 was \$32,854,000 and was itself greater than in any previous year. The increase in 1916 was therefore over \$18,000,000, or more than 54 per cent. Although this enormous increase in output was in large measure due to the large tonnage and high price of copper, yet nearly all other minerals were produced in greater quantities than in the previous year. During 32 years of mining Alaska has produced over \$351,000,000 worth of gold, silver, copper, and other minerals.

It can not be expected that Alaska will continue to produce so much mineral wealth each year, yet the large amount of preparation made in 1916 for lode and placer mining and the development of the coal-mining industry, now assured, give promise of a continuous healthy growth of the mining industry of the Territory. This is especially true of the Pacific coast region and of the territory served by railroads built or under construction.

The Alaska mines are believed to have produced gold to the value of about \$17,050,000 in 1916, compared with \$16,700,000 in 1915. This makes the total value of the gold mined in the Territory about \$277,900,000, of which sum \$196,800,000 has been won from the placers. In 1916 about 120,850,000 pounds of copper was produced in Alaska, valued at about \$32,400,000. The production of 1915 was 86,500,000 pounds, valued at \$15,100,000. The total copper production to date is 340,700,000 pounds, valued at \$68,300,000.

The value of Alaska's lesser mineral products in 1916 was about as follows: Silver, \$950,000; tin, \$120,000; lead, \$110,000; antimony, \$60,000; tungsten, \$50,000; coal, \$30,000; petroleum, marble, gypsum, etc., \$130,000.

PLACER GOLD.

The data in hand indicate that the value of the placer gold output in 1916 was \$10,640,000; in 1915 it was \$10,480,000. About 640 placer mines were operated in 1916, employing some 4,600 men. All the older districts appear to have held up or increased their output compared with the previous year, except Fairbanks. The increased output is, however, to be credited chiefly to the new camps of Marshall and Tolovana.

Thirty-six gold dredges were operated in Alaska in 1916, one more than in 1915—29 in Seward Peninsula, three in the Iditarod, and one each in the Ruby, Fairbanks, Circle, and Yentna districts. Of these 36 dredges four were installed in 1916. It is estimated that these dredges produced between \$2,000,000 and \$2,200,000 worth of gold. If the final figures bear out this estimate, it indicates a lower recovery per dredge than in the previous year. In 1915 the 35 dredges mined \$2,330,000 worth of gold.

LODE GOLD.

About 25 gold-lode mines were operated in 1916, compared with 28 in 1915. The value of this gold output increased from \$6,069,000 in 1915 to about \$6,200,000 in 1916. Southeastern Alaska, especially the Juneau district, is still the only center of large quartz-mining developments in the Territory. Next in importance is Willow Creek lode district. There was also considerable gold-lode mining on Prince William Sound, but a very decided falling off of this industry in the Fairbanks district. Lode-mine owners of Fairbanks are awaiting the cheapening of operating costs, especially of fuel, which will be brought about by the Government railroad.

COPPER.

The enormous copper output from Alaska mines in 1916 has already been referred to. During the year 18 copper mines were operated, compared with 13 in 1915—seven in the Ketchikan district, eight in the Prince William Sound district, and three in the Chitina district. The enormous output from the Kennicott mine, in the Chitina district, overshadowed all other operations. Had the transportation companies and smelters been able to handle the ore, however, many of the smaller copper mines would have made a much greater output. It is estimated that about 550,000 tons of copper ore was hoisted in 1916.

TIN.

About 232 tons of stream tin was produced in Alaska in 1916. Of this about 162 tons came from the York district, where two tin dredges were operated and a third was working on placer ground carrying both tin and gold. Developments were also continued on the Lost River lode-tin mine. The rest of the concentrates were recovered incidentally to placer-gold mining in the Hot Springs district of the lower Tanana Basin.

ANTIMONY.

The mining of antimony ore (stibnite) began in Alaska in 1915 and continued in a small way through the first half of 1916. The fall in the price of antimony during midsummer put an end to most of these operations. About 1460 tons of crude ore was mined and shipped during 1916. Much the larger part of this came from the Fairbanks district.

TUNGSTEN.

Though scheelite has long been known to occur in some of the Alaska placers, up to the last two years the demand for it has not been sufficient to encourage its recovery. The recent high price of tungsten has induced Alaska miners to turn their attention to scheelite deposits. In the fall of 1915 a scheelite-bearing vein was discovered in the Fairbanks district, and its development began. Later two other scheelite-bearing veins were found in the same district. During the winter some of these scheelite ores were treated in a local mill, and the concentrates were shipped out by parcel post. Scheelite mining was continued during the summer, and the crude ore was shipped out by steamer. Considerable scheelite was also recovered from some of the gold placers at Nome, and a little was produced in other districts. It is estimated that about 50 tons of scheelite concentrates were produced in Alaska during 1916, for which the producers received over \$50,000.

MINERAL FUELS.

The production of petroleum from the only oil claims patented in Alaska, in the Katalla district, continued in 1916. The operating company was reorganized and more extensive exploitation was undertaken. About 8,000 tons of coal was mined in Alaska during 1916 from half a dozen small mines. The largest producer was the Bluff Point mine, on Cook Inlet, where a lignite bed was exploited for the local market. The mining of coal in the lower end of the Matanuska field, for the use of the Alaska Engineering Commission, was also a significant event. This part of the field is already made accessible by the Government railroad, now under construction. The construction of a private railroad from Bering River into the Bering River coal field was also begun, and a little coal was mined at the southwest end of the Bering River field. Tenders for leases of coal lands in both the Bering River and Matanuska coal fields under the new law have been received by the Interior Department. Another impor-

tant event was the completion by the Geological Survey of a detailed examination of the more accessible part of the Nenana coal field, lying about 60 miles south of Fairbanks. All these facts indicate that systematic exploitation of the Alaska coal fields will soon be undertaken.

The figures received by the Federal inspector, as shown under the title "Coal mining in 1916" (p. 8), indicate that the production of coal may be somewhat higher than the estimate above.

WORK OF THE FEDERAL INSPECTOR.

The inspector was called to Washington, D. C., early in the spring for conferences regarding the leasing of coal lands in Alaska and while there completed a report on the leasing units and Government reservation in the Matanuska Valley, the field work for which had been done the previous summer and fall. While in Washington the inspector attended the National Safety-First Exhibit, February 21 to 26, and also the conference of State and Government officials on the standardization of mining statistics and mine regulations.

During the year two reports were prepared on mine inspection and the mining industry in the Territory. The first report, made to the Director of the Bureau of Mines, covered the calendar year 1915.^a It contained the new laws passed by the Territorial Legislature in regard to mine inspection, descriptions of mines throughout the Territory, lists of operators, tables of accidents, and recommendations for the advancement of the mining industry. The second report, made to the Secretary of the Interior, was for the fiscal year ended June 30, 1916. It reviewed in brief the mining industry and made certain recommendations for improvement, but did not go into the details of mining.

Early in the spring of 1916 the Federal and Territorial inspectors agreed to cover different parts of the Territory to avoid duplication of work and to obtain the greatest efficiency. Accordingly, the Federal inspector began his field work in the districts tributary to Cook Inlet and covered the country from there to Ketchikan. This area embraced the Willow Creek (Susitna drainage), Matanuska, Cache Creek, Crow Creek, Kenai Peninsula, Prince William Sound, Copper River, Juneau, Sitka, and Ketchikan districts. The inspection covered the operations of dredges, mills, metallurgical works, coal, lode and placer mines, the field work occupying most of the time from May to the latter part of December.

Before the Federal inspector returned from Washington, the Territorial inspector visited the districts of southeastern Alaska and went into the interior of the Territory as soon as navigation on the rivers opened. He visited the placer fields tributary to Fairbanks,

^a Smith, S. S., *The mining industry in Alaska during the calendar year 1915*: Bull. 142, Bureau of Mines, 1917, 65 pp.

Tolovana, Hot Springs, Ruby, Iditarod, and the Seward Peninsula. Leaving Nome on one of the last boats in the fall, the Territorial inspector made a trip to the west, visiting the Prince William Sound and Copper River districts.

NEEDS OF THE INSPECTION SERVICE.

HEADQUARTERS.

The headquarters of the Federal inspector are at Juneau, where he occupies, through the courtesy of the Department of Justice, the grand jury room in the courthouse when that body is not in session. However, the district court has ruled that criminal cases coming before the United States commissioner must be tried by jury, and, as the grand jury room is the only room large enough to accomodate the number of persons necessarily present at such trials, it becomes incumbent for the mine inspector to move out whenever a criminal trial is held before the commissioner. This procedure leaves the files and library of the inspector open to any who care to take advantage of the opportunity. Valuable books and bulletins have been removed from the office, or disfigured with pencil notations. The room is entirely inadequate for the present needs of the inspector's office and permanent quarters should be procured at an early date, especially as the opening of the coal mines throughout the Territory will greatly increase the work to be done. The coal mines in the Territory are operated on ground leased from the Federal Government and the duty of inspecting these mines rests on the Government, as the territorial mine inspection act specifically excepts coal mines from the jurisdiction of the territorial inspector.

An appropriation should be made to cover the rental of permanent quarters and provide funds for the purchase of office furniture and supplies. An appropriation of \$600 for office rent is recommended.

CLERICAL ASSISTANCE.

During the past year the Federal inspector has been able to keep the records of his office current and to attend promptly to all correspondence through the aid of Roy A. Dye, who was appointed clerk to the mine inspector in the fall of 1915 and has made his headquarters in Juneau. In addition to his office duties Mr. Dye has spent considerable time in training the miners near Juneau in first aid to the injured.

FIELD ASSISTANTS.

It would require a period of at least four years for one man to inspect all the mining operations in the Territory, and while he was

spending a season in one district the mines visited the year before in another locality would have ceased to exist and others would have been opened. The Federal mine inspector should have at least four assistants. And even with this number many mining operations in the more remote portions of the Territory could not be visited more than once annually and inspection based on one visit per year certainly can not be considered efficient.

Under present conditions the percentage of serious and fatal accidents that the inspector can investigate is extremely small and, owing to the time consumed in travel and communication, it may be impossible for the inspector when he finally arrives to determine the causes accurately. With the facilities now at hand the inspector can not investigate a tenth of the serious and fatal accidents and it is necessary for him to rely upon the statements of the operators, supplemented by those of employees, for at least nine-tenths of the information regarding the causes of accidents.

The infrequent visits of the inspector work a hardship both on the employee and operator, as the inspector often fails to see the mines under average conditions. To be specific, many of the mines have excellent ventilation in the winter, when conditions for ventilation are most favorable; but in the spring and fall, when atmospheric conditions are changeable and when the temperature outside varies but little from that underground, the relative humidity becomes so high and the air so contaminated that it is impossible for men to accomplish more than a small part of their normal work. If a mine is visited at this time the ventilation is condemned, whereas if the inspector had gone through the same mine a few months before he would have found the ventilation excellent.

The Federal mine inspector should have assistants enough to inspect thoroughly every mine of any consequence several times during the year and to investigate practically all the serious and fatal accidents. The visits should be so frequent that the inspectors would see all the mines under average rather than under exceptional conditions.

Many of the coal beds in the Bering River and Matanuska fields are extremely gaseous and great caution should be used in opening mines, whether under lease or free-use 10-acre permits. The inspector's office should be equipped with sufficient mine-rescue apparatus in each coal field to train the miners in its use and provide against the emergency of an explosion until the operators in the fields have sufficient apparatus and trained men to insure prompt action in case of accident.

MINE EXPERIMENT STATIONS.

Alaska has been particularly fortunate in having mine experiment stations, under the Federal Bureau of Mines, established at points that will serve both the coast and the interior districts. The station at Fairbanks will be able to handle such problems as may arise in the interior, and work along the coast will naturally go to the Seattle station. The Alaskan problems presented to these stations will be numerous, but particular attention will be given those that confront the pioneer. The metallurgy of the copper-bearing magnetites of the Ketchikan district, the development of electric smelting to utilize the vast water power available along the coast, the concentration of tungsten minerals, the metallurgy of tin ores, the economic recovery of minerals other than gold from placer gravels, the cleaning and coking of the bituminous coals with attendant by-product problems, the possibility of briquetting the lower-grade lignites, and a general dissemination of knowledge concerning the value and occurrence of many of the rarer minerals are only a few of the questions that will arise immediately. An effort to assist the prospector will do more to develop the Territory at this particular time than any other line of endeavor.

COAL MINING IN 1916.

The most important event in the Alaskan coal industry in 1916 was the announcement by the Government of the reservations and leasing units in the Matanuska and Bering River coal fields and the promulgation of rules and regulations governing the operations of mines under the coal-leasing act. Already a number of bids have been received, ranging up to \$2,000,000. These have been advertised in various periodicals and newspapers for the required period and the proper length of time has been given for competitive bidders to place their final statements. Owing to the time necessary for this work, it has been impossible to start active development on any of the leases this year, so the only coal development has been on patented areas and on the free-use 10-acre permits issued by the Government.

The Alaskan Engineering Commission has completed the Government railroad to the central part of the Matanuska coal field, and private interests have undertaken the construction of a standard-gage road from a point known as Goose City, on Controller Bay, to the Bering River field.

The free-use 10-acre permits, which apply to all coal land outside the surveyed areas in the Bering River and Matanuska coal fields, have been freely used to supply coal for local needs, and except for the clause restricting shipments to local points some coal would have been shipped to the States from areas held under them. The permits

provide that the applicant may begin mining coal as soon as he has mailed his application for the permit and may continue work for two years unless his application is denied by the General Land Office.

Under these conditions William Martin made application in the early spring to mine coal on Moose Creek, in the Matanuska field. Before he could be advised that this area constituted a part of one of the leasing units his foreman had driven a short drift and taken out about 100 tons of coal, which was hauled to the Willow Creek district, about 20 miles, and used for domestic purposes in the camp of the Alaska Free Gold Mining Co., which is under lease to Mr. Martin. Later a contract was let to Oliver La Duke to supply the Alaskan Engineering Commission with 1,000 tons of coal. His men entered the workings made by Mr. Martin's foreman and stoped the coal to the surface, leaving the ground in exceedingly poor shape for anyone who should lease it at any future date.

Later R. G. Doherty was given a contract to supply the commission with 2,000 tons of coal. Mr. Doherty, who had discovered a coal bed on the west bank of lower Moose Creek, about a mile above the railroad right of way, opened a mine described elsewhere in this report, and later received a free-use 10-acre permit covering the ground, in order that he might mine and sell to the local trade. This ground, although within the surveyed area in the Matanuska field, was not a part of any of the leasing units, and so did not conflict with the areas offered for lease. Under a second contract Mr. Doherty furnished the commission with several thousand tons and shipped about 300 tons to the various towns for local consumption, making a total for the year of approximately 6,000 tons.

The producer ranking next in point of tonnage was John A. Herbert, who obtained a permit to mine coal near Bluff Point, about 14 miles north of Seldovia, on Cook Inlet. Here a 6-foot seam of lignite, dipping approximately 8° , was opened by a drift on the bed and an air shaft for ventilation. Both walls are sandstone. The coal is fairly clean and a good domestic fuel. The property is nongaseous, and only eight men have been employed underground. The coal was shipped to Anchorage, Seward, and Seldovia, the total output being 1,500 tons.

The Happy Valley Coal Co., also operating near Bluff Point, made a small output, which was shipped to Anchorage. The company applied for a lease that it might ship coal to the States.

The Cache Creek Dredging Co. obtained a permit to mine coal on an area near its dredging ground, on Cache Creek, in the Yetna Valley, and mined about 20 tons of coal a day during the dredging season. The coal was used at the camp on the dredge. Most of this coal was taken from an open cut, but a contract has been let to drive

entries and an air shaft this winter, so that a supply of better coal will be available nearer the dredge early in the spring.

A small amount of work was done by Harry Dugdale, on the patented ground of T. P. MacDonald, in the Bering River field, and operations were resumed by the Alaska Petroleum & Coal Co. at its patented ground on Canyon Creek. Each of these areas contains 160 acres.

As the ground covered by permits is widely scattered and mining is intermittent, an accurate return on the amount of coal mined during the year is not possible, but probably the total production approximated 10,000 tons. No coke was manufactured nor was any coal shipped out of the Territory.

The following table shows the scattered areas covered by the free-use 10-acre permits:

Ten-acre permits for mining coal.

Operator.	Locality.
Raymond W. Silver.....	South shore of Knik Arm, on Cook Inlet.
I. D. Nordyke.....	Kachemak Bay, Cook Inlet.
Stimson & Heffner.....	Eight miles north of Susitna.
Ralph V. Anderson.....	Kachemak Bay, Cook Inlet.
George A. Esterbrook.....	Chicken Creek, Forty Mile.
Robert Brown.....	Watershed between Koyuk and Kewalic Rivers (north of Nome).
George Wallin.....	Kugruk River between Montana and Reindeer Creeks (north of Nome).
George F. Hendricks.....	Kootznahoo Inlet, Admiralty Island.
Cache Creek Dredging Co.....	Cache Creek, Yetna Valley.
John A. Herbert.....	Bluff Point, Cook Inlet.
Frank Wells, A. O. Wells, and Coal Creek, Chulitna Valley. John Coffey.	
Wm. Knox and Chas. Krutsinger..	Iditarod.
J. R. Uhl.....	Eight miles north of Anchor Point, Cook Inlet.
R. G. Doherty.....	Moose Creek, Matanuska Valley.
Fredrick Brassell.....	Lower end Matanuska field, just outside surveyed area.
Robert M. Aistrop.....	Shore, Cook Inlet, 24 miles south of Kaisilof River.
William Maitland and S. L. Carpenter McLennan.	Creek, Matanuska field, south of surveyed area.

GOVERNMENT RAILROAD.

The past season has been a busy one for the Alaska Engineering Commission which is building the Government railroad from the coast to the interior. The docks at Seward were rebuilt after being destroyed by fire during the spring, and the old line of the Alaska Northern has been rehabilitated, so that it is possible to get trains to Kern Creek although there still remains much bridge building and trestle work to be completed on this end of the line. Rails have been laid about 15 miles southeast of Anchorage to Potter Creek,

on Turnagain Arm, and a number of crews have been employed the past winter on rock work between this point and Kern Creek. Track has been laid to the central part of the Matanuska field and will now be advanced along the main line or by spurs, according to the greatest needs of the coal operators. From Matanuska Junction some 30 or 40 miles of track have been laid along the main line, and in the Susitna Valley between Montana Creek and Indian River over 70 per cent of the right of way has been cleared. The work at Nenana has proceeded up the Nenana to unite with the line coming up the Susitna at Broad Pass and up the Tanana toward Fairbanks. During the active part of the season the commission employed approximately 5,000 men. Coal has been shipped from Moose Creek to Anchorage, the freight rate being 75 cents a ton in carload lots. The passenger rate is 6 cents a mile.

ROAD AND TRAIL BUILDING.

In a pioneer country the maintenance of existing roads, the prompt building of trails to newly discovered districts, and the conversion of these trails into roads as the districts advance are of the utmost importance. There are three sources of revenue for the construction and maintenance of roadways throughout the Territory: First, the appropriation made by Congress to be expended under the direction of the Alaska Road Commission, which is managed by engineers from the United States Army; second, 65 per cent of the Alaska fund, composed of the revenues derived by the Federal Government from business and trade licenses outside of the incorporated towns; and, third, 75 per cent of the moneys which the Territory derives from the sale of timber on the National Forest Reserves. All told, these sums are usually less than \$250,000 per annum and are inadequate for a Territory as large as Alaska.

LABOR.

As a whole the supply of labor has not been adequate, especially during the late summer and fall. In the spring there was a slight excess of labor at the localities where railroad building was most active, but this was soon absorbed by the construction work. The high wages paid to miners in the States made serious inroads on the labor supply along the coast where the wages are comparatively low. Many properties were short-handed and the type of labor available was far from satisfactory.

Wages in Alaska are low in view of the cost of living and the short season of the interior. Machine men are paid \$3.50 to \$4 an 8-hour shift along the coast, the cost of board usually amounting to \$1 or \$1.25 a day, with extra fees for room, medical attention, etc.

Throughout the interior and on Seward Peninsula the average wage is \$5 per day and board, the hours varying somewhat in the different districts, though most of the underground work is on an 8-hour basis. In the interior, board is figured at \$2 to \$2.50 per day and on the Seward Peninsula at \$1.25 to \$1.50 per day. As most of the placer mines operate less than six months of the year, and as the wages then earned must be averaged against a long period of inactivity, \$5 a day and board is poor remuneration.

FIRST-AID AND MINE RESCUE TRAINING.

First-aid and mine rescue training represent a difficult problem for the mine inspector's office. The actual work of inspection is so much greater than what one man can accomplish that little time is left for giving men this much-needed training. During the stormy season in the winter, when travel is difficult, the inspector has utilized as much time as possible in assisting in training the miners of the Juneau district in mine rescue and first-aid work.

The following report of the first-aid contest on July 4, 1916, is taken from *The Gold Bar*, the official safety magazine of the Alaska Treadwell Gold Mines Co.:

At 8 o'clock the next morning (July 4) the second annual first-aid contest was held on Treadwell Field, with entries from the Treadwell, Mexican, Ready Bullion, and Alaska-Juneau mines. The contest was judged by Drs. Sloane, of Juneau; Sargent, of Douglas; Midford and Ellis, of Treadwell. The two-man and two-team events were given. First place was won by Alaska-Juneau, under the leadership of Capt. Monte Benson, with a score of 99 per cent. Treadwell, captained by C. R. Horner, and the Ready Bullion, captained by John Olsen, tied for second place, both scoring 96½ per cent. The Mexican, captained by Charles Hall, was close in their wake with 95½ per cent. All the teams are to be complimented on their excellent work, especially as each score sheet bore this legend: "When in doubt, dock."

Aside from its humane aspects, first aid has amply demonstrated its cash value in this district. To state accurately the amount of this saving is impossible, as no one can determine the time an injured man would have spent in the hospital had he not received first-aid treatment. As the Territorial act directs that the compensation paid by the employer varies directly with the length of time the injured man is incapacitated, it is greatly to the advantage of the employer to see that the latter is given the best of treatment.

The Territorial act fixes the employer's liability for fatal accidents to married men at \$3,000 and \$6,000, the amount varying with the number of dependents. On this basis the saving of one life represents a minimum saving of \$3,000. The records show that recently several men overcome by powder fumes have had their lives saved by the prompt application of artificial respiration. During the past five years the application of artificial respiration has saved the district immediately tributary to Juneau at least \$50,000 in direct sums.

Had these men died the community would have suffered an economic loss, and the operators would have had to pay out large sums in compensation.

The need of mine rescue apparatus and training may not be so apparent to the casual observer as the need of training in first aid. However, in the low-grade gold mines of the Juneau district poor ventilation in the long raises has been responsible for many deaths that might have been prevented by the proper use of self-contained breathing apparatus. As a specific example, the accident to Ole Roen (Table 1, p. 62) in the Alaska-Juneau mine, on September 6, 1916, may be cited. Roen was working for a contractor who was driving several inclined raises, and with his partner, Ernest Larson, went on shift shortly after 7 o'clock in the evening. The compressed-air pipe, the only means of ventilating the raise, was blocked by some dirt in the nozzle, and the raise was full of powder gases from the blasting of a previous shift. When the partners were about 160 feet up the raise the air became so bad that they turned back, but Roen became unconscious when he reached the 100-foot point and slid onto the footwall with his feet against the ladder. Larson reached the foot of the raise and notified several men, who tried again and again to reach the unconscious man. Their efforts were successful after three of them had been overcome by gas, one falling into the waste pocket and injuring his hip. All the unconscious men were given artificial respiration, but Roen had been in the gas too long and never regained consciousness. The total result of this accident was one man killed, one in the hospital with an injured hip and suffering from the effects of gas, two others in the hospital suffering from the effects of the gas, and a temporarily disorganized crew. All this could have been prevented by a properly-trained man wearing breathing apparatus. The financial loss in this one accident would have more than covered the cost of equipping the mine with such apparatus.

At practically all of the large mines throughout the Territory competent physicians are employed, and at some, notably the Alaska Treadwell, an effort has been made to drill the men in first-aid methods. These attempts, however sincere, are more or less spasmodic.

MERIT RATING FOR MINE INSURANCE.

For several years the companies operating the large low-grade gold mines of the coast have enjoyed a particularly favorable insurance rate (\$2.75 per \$100 pay roll) in spite of the fact that the Territorial compensation is larger than the average paid in the States. At the close of 1916 two of the largest mines had their insurance withdrawn by the insurance companies on account of the number of accidents, and were offered new insurance at a rate that would

have increased their premiums nearly \$50,000 annually. It is not possible to say that the original rate could have been retained, but the higher rate represents a material loss to the stockholders. Had the men been carefully trained in first aid, had the mines been equipped with rescue apparatus, and had a competent safety engineer been empowered to carry out a comprehensive plan of accident prevention, it is probable that the corporations would still be enjoying a favorable insurance rate.

The merit rating of mines for insurance rates has been adopted in the larger mining centers of the East, arrangements are being perfected for its introduction in California, and, sooner or later, it is bound to come to Alaska. The accident ratio in Alaska is fairly high and there is no reason why it can not be materially reduced if the operators give more attention to the study of accidents. A careful investigation of all accidents would soon lead to measures for their prevention and automatically reduce the rate to a point where the corporations would be warranted in applying for more favorable insurance rates.

COST DATA.

Conditions in the Territory are so variable that mining costs have a wide range. The different items vary with the seasons, and in order to reduce costs to the minimum the outlay for mining and for the transportation of supplies must be spread over a period of years in order to take advantage of climatic conditions.

The record for low costs for lode mining in the Territory is held by the Alaska Gold Mines Co., its annual report giving \$0.69039 a ton as the total cost of mining and milling. These figures are possible only at a property favorably situated with regard to transportation and labor conditions and handling an extremely large tonnage. At several of the best-managed lode mines in the Fairbanks district the costs range between \$15 and \$20 per ton.

In estimating costs in placer mining the customary unit of measurement is the square foot of bedrock cleaned. On the Seward Peninsula the cost of working the drift mines in frozen ground ranges upward from 35 cents a square foot of bedrock, according to the nature of the ground and the location of the claim. Near Fairbanks it is somewhat higher. On a claim typical of the deep ground of that district, the cost, including overhead expense, amounted to 90 cents a square foot. Here the bedrock was a comparatively soft schist, the ground was frozen, the gravel of medium size with no large boulders, and the depth about 180 feet. In working the ground about 1½ feet of bedrock was taken up and 3 or 4 feet of gravel was mined.

About the same conditions govern dredging costs, though much depends on whether the gravel is frozen or not, and in frozen ground

the cost may vary considerably because of careful or careless thawing. On the Seward Peninsula, where crude oil is available for fuel, thawing costs $12\frac{1}{2}$ to 15 cents a cubic yard. In the Iditarod, where wood is used for fuel, thawing costs may range from $37\frac{1}{2}$ to 60 cents a cubic yard. The cost of dredging, including amortization and depreciation but not including thawing, for the entire Territory will probably average approximately 25 cents a cubic yard or more.

To insure maximum efficiency in handling frozen ground the thawing points should be carefully placed with regard to the kind of material to be thawed. Also the thawed ground should be allowed to stand a month or two before the dredge reaches it to assure the full benefit of the heat.

The principal items of cost are labor, food, fuel and freight. Miners in southeastern and southwestern Alaska receive \$3.50 to \$4.50 a day; in the interior and on the Seward Peninsula, \$5 a day and board. The costs of boarding men in southeastern and southwestern Alaska is figured at \$1 to \$1.25 per day; on the Seward Peninsula, at \$1.25 to \$1.50; and in the interior, at \$2 to \$2.50; but it necessarily varies with the size of the crew.

In southeastern Alaska, companies buying comparatively large lots of fuel pay about \$6 per ton (2,000 pounds) for coal and \$1 a barrel for oil at tidewater; on Prince William Sound, \$9 a ton for coal and \$2 a barrel for oil; at Nome, \$15 a ton for coal and \$2.50 a barrel for oil; at Fairbanks wood is \$12 to \$16 a cord delivered. Such prices will undoubtedly be materially lower when the Government railroad is completed and coal and oil from Alaskan fields are available.

In the following tables of freight rates, 2,000 pounds or 40 cubic feet are considered a ton. It is the practice to charge by weight or by bulk, according to which basis will yield the greater profit to the carrying company. On extra heavy pieces an excess charge is made. Full details of local freight rates can not be given because of variations at different places and in different seasons, but the following tables indicate the rates to the larger centers:

Freight rates, per ton, from San Francisco, Seattle, or Tacoma.

[Ton—2,000 pounds or 40 cubic feet.]

Destination.	General rate.	Coal.	Mining machinery. ^a	High explosives.
Andreaski.....	\$28.00	General rate.	General rate.	General rate plus 100 per cent.
Marshall.....	30.00	do.....	do.....	Do.
Dikeman.....	50.00	do.....	do.....	Do.
Ruby.....	42.00	do.....	do.....	Do.
Fairbanks.....	55.00	do.....	do.....	Do.
Rampart.....	50.00	do.....	do.....	Do.
Circle.....	55.00	do.....	do.....	Do.
Eagle.....	57.00	do.....	do.....	Do.
Bettles.....	80.00	do.....	do.....	Do.

^a Extra charge for pieces weighing over 2,000 pounds.

Freight rates, per ton, for carload lots, from Seattle or Tacoma.

[Ton—2,000 pounds or 40 cubic feet.]

Destination.	General rate.	Coal.	Mining machinery. ^a	High explosives.
Nome.....	\$15.50	\$12.00	\$15.00	\$32.00
St. Michael.....	15.50	12.00	15.00	32.00
Golovin ^b	16.50	8.50	10.00	27.00
Teller ^b	12.50	9.50	12.50	32.00
Deering ^b	14.00	9.00	13.00	35.00

^a No single piece weighing over 3,000 pounds, per ton, weight or measure.^b Does not include lighterage.*Freight rates, per ton, from Seattle or Tacoma.*

[Ton—2,000 pounds or 40 cubic feet.]

Destination.	General merchandise.	Coal.	Machinery. ^a	High explosives.
Ketchikan.....	\$6.50	^b \$3.00	\$6.50	\$12.00
Sulzer.....	9.50	^b 4.00	9.50	22.00
Wrangell.....	7.50	^b 3.00	7.50	17.00
Petersburg.....				
Douglas.....				
Juneau.....				
Treadwell.....	8.50	^b 3.50	8.50	22.00
Skagway.....				
Sitka.....	10.00	^b 4.00	10.00	22.50
Katalla Anchorage ^c	12.50	^d 7.00	12.50	20.50
Ellamar.....				
Cordova.....				
Latouche.....				
Valdez.....				
Seward.....	14.00	^d 8.00	14.00	23.00
Seldovia.....				
Knik Anchorage ^c	14.50	^d 8.00	14.50	23.00
Ilhiamia Bay.....	17.50	^d 10.00	17.50	35.50
Kodiak.....	16.50	^d 9.00	16.50	30.50
Appollo.....	16.50	^d 11.00	17.50	37.50

^a Extra charge for pieces over 4,000 pounds.^b In bulk on freight boats at owner's risk.^c Does not include lighterage.^d In sacks.**MINES AND DISTRICTS.****SOUTHEASTERN ALASKA.****JUNEAU DISTRICT.****ADMIRALTY ALASKA GOLD MINING CO.**

The company is developing a group of 52 lode claims on the south shore of Funter Bay, on Admiralty Island. As the claims had been worked previously by several organizations considerable work was done at scattered points. Several crosscuts have been driven to tap the ore bodies and a shaft sunk 110 feet. From one crosscut drifts were run for about 150 feet on the ore and the vein stoped to surface, approximately 75 feet. The ore was crushed in two Fulton five-stamp batteries.

The mill has been rehabilitated and the company plans to develop the known ore bodies and to drive a crosscut at the base of the hill to tap several veins exposed on the surface.

ALASKA GOLD MINES CO.

The Alaska Gastineau Mining Co., which works the Perseverance mine in Silver Bow Basin, about 4 miles from Juneau, is the operating company for the Alaska Gold Mines Co. The ore is in an enormous fissured zone of slate and metagabbro cemented by a network of quartz lenses and veinlets. The mine was first opened by what was known as the Gilbert workings, the present fifth level, and later by a 1,400-foot crosscut, known as the Alexander crosscut, approximately 1,000 feet below the Gilbert workings. When the Alaska Gold Mines Co. assumed control plans were made for extensive development. The shaft, with stations at every 200 feet, was sunk from the Alexander crosscut to the thirteenth level, and the development of the previously opened levels (200 feet apart) between the Alexander crosscut and the Gilbert workings continued. A 12,000-foot tunnel was driven from Sheep Creek to connect with the bottom of the shaft and a system of oreways, raises, and stopes driven to permit the rapid handling of an enormous tonnage.

The stopes are worked on a full-breast shrinkage system, just enough ore being drawn to give headroom for the machines. Pillars are left at varying intervals. The ore is cut out along the footwall of the stope and caves from there to the hanging wall with little additional blasting. From the stopes the ore passes over grizzlies into the chutes, the oversize being "bulldozed" in bulldoze chambers. From the stope chutes the ore runs into 4-ton cars, of the Granby self-dumping type, which storage-battery motors haul to the main oreways. From the oreways the ore goes on 10-ton cars, hauled by electric motors, to the mill 6,000 feet from the portal of the tunnel.

The mill was designed to treat 6,000 tons of ore a day, but is capable of handling 25 to 50 per cent more. The cars are dumped four at a time by a revolving tipple, the oversize from the grizzlies going through gyratory and jaw crushers to unite with the undersize in a 10,000-ton storage bin cut in solid rock. From this bin a belt conveyor takes the ore to the mill and a second conveyor distributes it to the ore bins. From the bins the ore passes to large rolls and impact screens, the oversize being returned by automatic self-dumping skips to the first set of mill bins and the undersize passing to a second set of bins to be drawn into smaller rolls and impact screens, also set in a closed circuit. The dry pulp from the last bins is concentrated on double-deck Garfield tables, reground in tube mills and passed to Wilfley tables, the concentrate going to the retreatment plant. One of the noteworthy features of the mill is the independence of each unit, the bins being so arranged that the stoppage

of one part of the mill does not affect another until the bins in between are empty.

Power for the mine and mill is obtained from several sources. At the mine a small hydroelectric plant is driven by the water of Gold and Survey Creeks, and a large reservoir and hydroelectric plant have been constructed on Salmon Creek, about 4 miles from Juneau. The dam is of a radial arch type, 165 feet in height and 720 feet in length along the crest. The reservoir permits the delivery of 6,000 hp. the year around. Another plant, now under construction on Annex Creek, a tributary of Taku Inlet, will have an initial capacity of 4,000 and an ultimate capacity of 12,000 hp.

ALASKA GOLD BELT MINING CO.

The company is developing the Nelson-Lott group of claims, at the head of Sheep Creek Basin, about 5 miles southeast of Juneau.

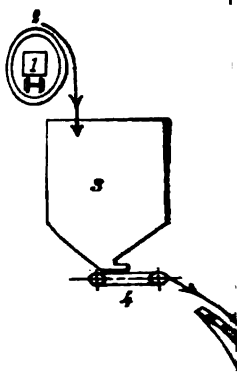
ALASKA JUNEAU GOLD MINING CO.

The Alaska Juneau mine is in Silver Bow Basin, about 2 miles from Juneau, and is under the same management as the Treadwell mines on Douglas Island. The ground embraces 29 patented and 88 unpatented lode claims as well as 24 mill sites.

The ore lies in a shear or fracture zone that forms an unusually broad lode but is cut by the Silver Bow fault so as to make two main ore bodies, known locally as the north and the south. The shear zone passes through slate and metagabbro and is traversed by a network of quartz lenses and stringers; its average commercial width is about 400 feet, but its limits have not been determined. It strikes N. 50° W. and dips 60° NE. The strike of the Silver Bow fault is N. 80° E., the dip 70° NW., and the throw approximately 2,000 feet.

In early development work the ore was mined in a glory hole on the side of the mountain, drawn through an adit below and treated in a 30-stamp mill that could run only a short time in summer months, because of the amount of snow in the pit and the scarcity of water. When the grade of the ore proved to be satisfactory, the Gold Creek tunnel was driven to tap both ore bodies and connections were made with the surface; subsequent development was from these openings. This tunnel with its branches constitutes the main haulage way, and is designated as the fourth level, its portal having an elevation of 450 feet. From the portal, a tram, which passes through several short tunnels, has been constructed along the hillside above Juneau and the ore from the development is treated in a pilot mill just south of the city. In 1916 contracts were let for a mill with a capacity of 6,000 to 8,000 tons in 24 hours. This mill, on a site adjacent to the pilot plant, will be completed early in 1917.

BUREAU OF MINES





The south ore body is opened by the No. 1 raise from the fourth level with the third and second levels above, the difference in elevation between levels being 250 feet. The raise starts 6,600 feet from the portal of the tunnel, is 760 feet long, and is driven upward on a slope of 50°. On No. 2 level drifts and crosscuts have been run to explore the ore body and connections are made with the surface. A little stoping has been in progress, but for the present the bulk of the mill feed from the south ore body will come from the glory hole on the surface. The ore ways extend from the cut to the fourth level with bulldozing chambers under the pit, where all large pieces of rock will be broken before they enter the main chutes.

On the north ore body No. 2 raise was driven from the fourth level and connection made to the surface by a 350-foot adit. From this raise the intermediate third, second, first, 0, and 00 levels have been driven. The raise, 8 by 14 feet in the clear with two compartments, is 1,520 feet long and slopes upward 56°. The main levels are all 250 feet apart vertically, the intermediate being 50 feet above the fourth. On the second, third, intermediate, and fourth levels drifts have been driven roughly paralleling the fault and from 100 to 150 feet apart. On the fourth level are chutes with bulldozing chambers at the top every 42 feet to the intermediate; from the chutes are openings into the main stope which is approximately 240 feet wide and 800 feet long. These chutes are fitted with 6-foot arc gates and will be covered with heavy grizzlies to prevent large slabs from falling in and blocking them. Above the intermediate level are raises approximately 100 feet apart and roughly parallel to the Silver Bow fault and the footwall that extend to the third and second levels. From these raises are driven the powder drifts, 30 to 50 feet long and differing about 30 feet in elevation. Several of these drifts just above the intermediate level have been shot with excellent results, and unless some unforeseen difficulty appears, this method of breaking the ore will be used to the surface.

Direct current at 550 volts is used for all electric haulage. At present this current is purchased, but an oil-burning steam plant under construction on the beach will soon be ready to deliver current. The distance from the portal of the Gold Creek tunnel to the mill is 6,200 feet and the trains over the main line will be made up of thirty or forty 12-ton cars. A 30-inch track gage is used, with 50-pound rails on the main and 40-pound rails on the branch lines. The haulage motors comprise three 9-ton General Electric, one 5½-ton Jeffrey, two 18-ton Westinghouse, each made up of two 9-ton articulated units, and two 5½-ton storage-battery locomotives.

Plate I shows the flow sheet of the 8,000-ton mill now under construction. This flow sheet, of course, will be subject to any modifications that may be demanded when crushing begins.

ALASKA PEERLESS GOLD MINING CO.

The company is opening a group of three lode claims, about 2 miles from the beach at Windham Bay. The ore, which strikes S. 70° E. and dips approximately 67° SW., has been opened by two tunnels, one 70 and one 175 feet long. The company plans to put in an air compressor and a power plant this spring.

ALASKA TREADWELL GOLD MINING CO.

The Treadwell group of mines is situated on the northeast side of Douglas Island, across Gastineau Channel, and about 2½ miles south-east from Juneau. The group consists of four mines—the Treadwell, the Seven Hundred, Mexican, and, after a 2,000-foot interval, the Ready Bullion. The four mines are operated by three separate companies, but steps have been taken toward the consolidation of the different groups under one corporation and they may be considered as one enterprise. The Alaska Treadwell Gold Mining Co. operates the Treadwell mine, the Alaska Mexican Gold Mining Co. the Mexican mine, and the Alaska United Gold Mining Co. the Ready Bullion and Seven Hundred mines.

The ore on Douglas Island is in two distinct dikes of albite-diorite, a deep-seated intrusive rock related to the granites, that form the backbone of the coast range of mountains.

This rock has been considerably altered and is traversed by a network of quartz lenses and stringers. The dikes have a hanging wall of metagabbro, also called greenstone, and a footwall of black slate. The ore, which contains occasional horses of schist, has a general strike of N. 60° W. and an average dip of 60° NE. The larger of the two dikes forms the ore body of the Treadwell, Seven Hundred, and Mexican mines; the smaller, about 2,000 feet south-east of the Treadwell-Mexican dike, furnishes that of the Ready Bullion. Along the Treadwell dike the ore is 40 to 400 feet wide and along the Ready Bullion 90 to 270 feet.

At first the ore was blasted into glory holes, drawn out on levels below, hoisted to the surface, and sent to the mills. The varying width of the ore bodies necessitated resort to underground mining at different depths, so that the levels in the different properties were not driven at like elevations, but below the 750-foot level the Treadwell, Seven Hundred, and Mexican constitute practically one mine. A long drift on the 1,100-foot level connects these properties with the workings of the Ready Bullion. The Treadwell, Seven Hundred, and Mexican properties have been opened by vertical shafts with levels at intervals varying from 110 to 200 feet.

The Seven Hundred shaft, after being enlarged to 4 compartments 7 feet 2 inches by 22 feet in the clear, to the 2,700-foot level, was

renamed the Central shaft and a new hoist and coarse-crushing plant erected. During the past year the work of enlarging and sinking the Mexican shaft, which will be known as the Combination, began. This shaft, when completed, will be 7 feet 8 inches by 31 feet 4 inches over the timbers and have five compartments. From the Central shaft drifts have been started on the 2,500 and 2,700 foot levels, the lowest developed level at present being the 2,300-foot. From the 1,750 to a point below the 1,950-foot level is a lean streak in the ore body, and a 300-foot floor pillar has been left from the 1,750-foot level to the backs of the stopes on the 2,100-foot level. The only openings between these levels are the raises for ventilation. At the Ready Bullion a new 4-compartment shaft, driven upward at an angle of 70° from the 2,400-foot level, will not only save long crosscuts on the different levels but will facilitate hoisting and ventilation. Levels in the part of the mine now worked are all 200 feet apart vertically.

Mining is by the shrinkage system, or back-stopping in ore-filled stopes. Levels are driven from the shafts at intervals of about 200 feet vertically, and sublevels are driven 25 feet above. Bulldozing chambers are opened along the sublevels and the stopes the full width of the ore and from 60 to 100 feet wide are cut 25-foot pillars, in which the manways are driven between levels, being left between stopes. Chute raises are driven from the main to the sublevels; through the latter enough ore is drawn to keep the ore about 7 feet from the back. In the bulldozing chambers on the sublevels all the larger slabs are blasted for easy handling in the cars on the main levels. After a stope is finished the ore is drawn and the pillars allowed to cave. The ore is trammed from the chutes to pockets at the shaft in two-ton cars, hauled by horses, storage-battery locomotives, gasoline locomotives, or tail-rope systems. From the pockets at the several shafts the ore runs into skips; then it is hoisted to the surface, is dumped automatically over grizzlies, passes through gyratory crushers to ore bins, and goes thence to the various mills.

The fine-crushing plant consists of five mills with a total of 960 stamps, and has a capacity of approximately 5,000 tons of ore in 24 hours. The crushed ore is concentrated on Frue vanners, all amalgamation plates having been removed during the past year. The concentrates, about 100 tons a day, are treated in the cyanide plant which contains the bullion refinery. The ore fed to the stamps is ground to pass a 3-inch ring and is crushed to 20 mesh before going to the vanners. The stamps weigh 1,240 pounds, fall 100 times per minute, and drop 9 inches. At the cyanide plant the treatment comprises regrinding in cyanide solution in tube mills, agitation in

Pachuca tanks, filtering, precipitating the gold in filter presses, and refining the bullion.

Power is obtained from four sources. The original and cheapest source is water power from impulse wheels at the various mills and compressors. This water is collected on Douglas Island by a system of dams, canals, ditches and pipe lines, extending from Fish Creek, 14 miles northwest of Treadwell, to Ready Bullion Creek, 3 miles southeast. During the wet season the supply is over 4,000 horsepower. In 1910 the policy of electrifying the power system was begun and as a result the Sheep Creek hydroelectric plant, a flood-water plant having a capacity of 2,500 kilowatts, and the Nugget Creek plant, having a capacity of 3,000 kilowatts, were constructed. These plants are on the mainland, the Sheep Creek plant being 4 miles southeast of Juneau, and the Nugget Creek plant, at the foot of Mendenhall Glacier, 12 miles northwest of Juneau.

The fourth source is steam-generated electric power from a central power plant at Treadwell. This plant contains four generators, each directly connected to a steam turbine, running at 3,600 r. p. m. This plant is used when the power supply from the hydroelectric plants fails and has a capacity, when the power is taken at Treadwell, equal to both hydroelectric plants. Direct steam power is being eliminated wherever possible about the mines and mills. California crude oil is used under all boilers for generating steam power or heat. The oil is brought north in tank ships and is stored in eight tanks at Treadwell, which have a capacity of 180,000 barrels. The annual consumption of oil is a little more than 200,000 barrels.

Under the general supervision of the company, an association, known as the Treadwell Club, is maintained for the recreation of the men. A building 56 feet wide by 206 feet long contains a reading room supplied with newspapers and magazines from all over the world, a pool and billiard room, cigar and candy stand, bowling alleys, an auditorium with a seating capacity of 500, and lavatories. In addition there are a natatorium containing a swimming pool 30 by 70 feet, tub, shower, and steam baths, and an athletic field, tennis and handball courts. The dues of \$1 a month entitle the members to all the privileges of the club and free admission to the entertainments.

ALASKA TREASURE MINE.

Only assessment work was done during the year at this lode gold mine, locally known as the Nevada Creek property, on Douglas Island, about 3 miles southeast of Treadwell. In the early development a 20-stamp mill was built near the outcrop, but five of the stamps have been moved near the beach, where a long crosscut has been driven to the ore, and it is planned to use the five stamps to sample the ore on a large scale.

EAGLE RIVER MINING CO.

The Eagle River group of 24 lode-gold claims is at Amalga, about 30 miles north of Juneau and 7 miles from tidewater, on the Lynn Canal, to which a horse tramway has been constructed. The ore body consists of quartz lenses and stringers through a shear zone in a slate-graywacke series.

The old workings, consisting of 10 levels from the face of the mountain, have been abandoned, and the only development being done at present is on what is known as the flume level, 600 feet below the former opening. Here the ore has been explored by over 2,700 feet of drifts and crosscuts, and some stoping has been done.

The ore goes to the mill, 400 feet below the flume level, over a balanced tramway. The mill contains twenty 1,250-pound stamps, which drop 7 inches 95 times a minute. The ore is crushed to 20-mesh and passes over amalgamating plates to four Wilfley tables.

EBNER GOLD MINING CO.

Exploration work continues at the Ebner mine, which joins the Alaska Juneau, in Silver Bow Basin. On the main, or No. 0, level over 3,000 feet of crosscuts and drifts were driven during the year, making approximately 13,000 feet of development on that level. A 442-foot raise was completed between the main level and the old workings above.

A testing plant comprises a 5-stamp battery of 800-pound stamps, amalgamating plates, Wilfley table, and regrinding apparatus. Power for the plant and a 12-drill 2-stage Ingersoll-Rand compressor is furnished by a 4-foot Pelton wheel under a 480-foot head.

JUALIN ALASKA MINES CO.

The Jualin Alaska Mines Co. has taken over the options held by the Algonican Development Co. on the Jualin mine and adjoining claims, giving this corporation control of a group of 30 lode claims and a mill site.

Two veins, the north and back ore bodies, have been opened. These strike northeast and dip about 60° NW. The north ore body is 6 inches to 18 feet wide, averaging between 7 and 8 feet. The back vein is 6 inches to 4 feet wide, averaging 3 feet. Both veins have been opened by a crosscut on what is known as the adit level. From this level raises have been driven to the surface and shafts sunk on the ore bodies to the 160 and 310 foot points, from which the development has been carried forward. The ore is mined by overhand stoping, the walls being so strong that only stulls are used in timbering. The gangue is quartz and the country rock a blocky diorite. Two 8 by 10 inch and one 6 by 8 inch geared hoists driven by com-

pressed air are used for hoisting the ore to the adit level. Below the adit level the mine is drained by a 5-inch 3-stage Byron Jackson centrifugal pump, direct connected to a 150-horsepower motor.

At the mill the ore passes over a 1½-inch grizzly, the oversize going to a No. 2 Gates gyratory crusher. From a 50-ton bin it is automatically fed to ten 750-pound stamps, which drop 8 inches 100 times a minute. The screen is varied from 25 to 40 mesh, according to the grade of the ore, and the pulp is run over amalgamating plates to Isabelle vanners. The concentrate is shipped.

A 42-inch Pelton wheel under a 125-foot head supplies power. In winter, as the water diminishes, a 25-horsepower motor belted to the line shafting serves as an auxiliary. Electricity is developed at the main power plant by a 100-kilowatt Westinghouse generator, driven by a Pelton wheel under a 550-foot head. A second Pelton wheel is set directly on a line shafting belted to an Ingersoll-Rand two-stage compressor, with a capacity of 2,500 cubic feet of free air a minute. The power plant is so arranged that two semi-Diesel 150-horsepower engines can be belted to the main line shafting in periods of low water.

KENSINGTON MINING CO.

The company continued to explore its property in the Berners Bay district, about 60 miles northward from Juneau. The new mill, plans for which were formulated last year, was not built.

NORTHERN EXPLORATION CO.

The company is developing, under bond, the Enterprise and Montana groups, each containing five lode claims, on the north shore of Limestone Inlet, about 25 miles south of Juneau.

At the Enterprise group, a drift has been driven 320 feet on the vein at an elevation of 1,300 feet. A 77-foot raise connects this and a second drift, which is in 120 feet on the ore. The vein strikes N. 30° E., dips 52° NW., and averages about 30 inches in width. The country rock is diorite and the gangue quartz. Stopes have been opened on both levels and the ore is conveyed from the mouth of the lower drift to the mill on the beach by a jig-back tram, with a 1-inch track cable, ¾-inch haulage cable, and two buckets with a capacity of 500 pounds each.

At the mill the ore passes over a 1-inch grizzly to an 8 by 12 inch jaw crusher and is fed from the bins to the stamps by Challenge feeders. The battery is of Frazer & Chalmers design and contains five 1,250-pound stamps falling 6 inches 98 times per minute. Forty-mesh screens are used. The pulp passes over amalgamating plates to concentrating tables.

GRANBY CONSOLIDATED MINING, SMELTING & POWER CO. (LTD.).

The company has been working the Mamie and It mines during the year, besides doing considerable exploration of other claims in the locality.

The Mamie mine, at Hadley, on Lyman anchorage, on the coast of Prince of Wales Island, is opened by a 300-foot shaft and a crosscut which taps the shaft on the second level 170 feet below the collar. There are four levels at intervals of 95, 75, 60, and 65 feet. The ore is chalcopyrite in a silicious and magnetite gangue associated with small amounts of gold and silver. The ore bodies, irregular lenses of greatly varying size, are found in an altered zone between a granodiorite intrusive and limestone and greenstone. One of the lenses measures 5 feet by 20 feet by 20 feet; another 150 feet by 300 feet by 75 feet. These figures give no idea of the size of the magnetite bodies, as the ore has been mined only as far as the copper content made it valuable. The lenses have a north and south strike, approximately parallel to the contact, and are practically vertical. Because of the ore bodies varying in size, no one method of mining is applicable to the whole mine.

Part of the ore near surface was mined through a glory hole; underground some stopes have been worked on a full-breast system and in others the ore has been shot off in benches. All drilling, except that done by contractors, is on the day shift, and the ground blasted at night by a shot firer who handles all the powder. This plan eliminates much of the danger of handling powder and provides much better ventilation than is produced by indiscriminate blasting.

One 60-hp. return tubular and one 90-hp. locomotive type boiler furnish power. The ore is raised with a 12 by 18 inch two-cylinder, double-drum Bradley hoist. Leyner drills and jack hammers are used underground. The ore is conveyed from the mine to the wharf over a 7,200-foot Riblet tram, having a capacity of 300 tons in 9 hours. The diameter of the track cable on the loaded side is $1\frac{1}{2}$ inches, on the empty side 1 inch; the haulage cable is $\frac{3}{4}$ inch and the capacity of the buckets 7 cubic feet. There are 13 towers and the upper terminal, elevation 750 feet, has a bunker capacity of 700 tons. The lower bunkers hold 150 tons.

The It mine is situated on Kasaan Bay, about three-quarters of a mile from the beach. The ore is chalcopyrite, but differs from that at the Mamie in being higher grade and having a gangue of altered limestone. It lies in lenses between intrusive granodiorite and limestone. When the company took over the mine there was a 150-foot vertical shaft and a 1,500-foot crosscut, 280 feet below the bottom of the shaft. A 2-compartment raise was driven to the shaft and the ore passed down this to the tunnel level. From the mouth of the tunnel an 8-hp. gasoline motor hauls the ore to the beach,

where it is stored in bunkers, having a capacity of 1,200 tons. A large portion of the work this year has been diamond-drill exploration. The company furnishes living quarters at both mines, those at the Mamie being a little better equipped as regards change rooms and showers on account of the greater number of men working there.

JUMBO MINE.

The Jumbo mine, commonly known as the Sulzer, of the Alaska Industrial Co., has been leased the past year to Charles A. Sulzer. It is situated on Hetta Inlet, on the West coast of Prince of Wales Island.

The ore, mainly chalcopyrite, lies in lenses in an altered zone between granite and limestone. These lenses show little continuity and prospecting is difficult. The main entrance to the mine is a cross-cut at an elevation of 1,500 feet, though many of the ore bodies have been near enough to surface to be worked as open cuts during the summer. A 252-foot winze has been sunk from the crosscut and several levels, with connections to the surface for ventilation, opened on the ore bodies. A Hallidie hoist raises the ore from the winze to the crosscut level, whence it is trammed to the entrance of the cross-cut, and then sorted before going to the bins which constitute the upper terminal of an 8,000-foot aerial tram. The lower terminal is at the 4,000-ton bunkers on tidewater. The track cable has a diameter of $1\frac{1}{8}$ inches on the loaded and seven-eighths inch on the empty side; the haulage cable is five-eighths inch.

At the mine a Sullivan compressor, driven by a 60-horsepower Allis-Chalmers motor, furnishes air for the hoist and machines. The power comes from a hydroelectric plant on the beach where a 50-kilowatt General Electric dynamo is belted to a 36-inch Pelton wheel under a 360-foot head.

There is a sawmill on the beach and during the summer two new bunk houses, a superintendent's house, and a number of other buildings have been erected.

A deposit of barytes has been opened by Mr. Sulzer at Lime Point, several miles below Sulzer, and machinery is on the ground for the erection of a grinding plant in the spring of 1917.

MOUNT ANDREW MINING CO.

The Mount Andrew mine is situated 3,600 feet from tide water, on the north shore of Kasaan Bay, on the east coast of Prince of Wales Island.

The ore, chalcopyrite-magnetite lenses of irregular size and shape in an altered zone between a granodiorite and a limestone, is mined for its copper content. Large bodies of low-grade ore opened in the stopes and glory hole will be available for a concentration plant in the future.

The largest opening made to date is 40 feet wide and 230 feet long. Most of the mining has been through glory holes, though a recently completed crosscut about 1,600 feet long and a 310-foot raise tap the ore from below. Recently most of the development work has been confined to the main and a number of outlying ore bodies at the surface.

A 3,800-foot Riblet automatic tram, with a capacity of 300 tons in 8 hours, connects the mine and wharf. There are 16 towers and the buckets have a capacity of 1,100 pounds each. The diameters of the cables are: Loaded side, 1 inch; empty side, three-fourths inch; and haulage cable, five-eighths inch. Denver Rock Drill Co., Ingersoll-Rand, Sullivan, and Waugh machines are used in mining, the air being supplied by two 6-drill Ingersoll-Rand steam-driven compressors at the beach.

PRINCETON MINING & MILLING CO.

The Valpariso mine is situated on the shore of Paul Lake, near Dolomi, on the eastern side of Prince of Wales Island.

The vein varies in width from a mere seam to 14 feet and averages about 4 or 5 feet. The gangue is quartz and lime; the country rock, schist and dolomitic limestone. The strike is, roughly, east, and the average dip is 30° north.

A shaft has been sunk on the ore a little over 300 feet, with four levels at intervals of 90, 50, 50, and 128 feet. A drainage tunnel runs from the second level to the lake.

At the mill the ore passes over a 1-inch grizzly; the oversize is reduced to that size by an 8 by 10 inch Blake crusher and falls with the undersize into a 100-ton bin. A Challenge feeder feeds a five-stamp Hendy battery in which the stamps weigh 1,250 pounds each, and drop 9 inches 90 times a minute. The ore, crushed to one-fourth inch, passes to a 5-foot Chilian mill, which crushes it to approximately 80 mesh. The pulp goes to a classifier and amalgamation plates, the oversize returning to the Chilian mill. The plates, 5 by 12 and 5 by 16 feet, are set on a grade of 1 inch to 1 foot. From these the pulp passes to a Callow classifier; the coarser product goes to Diester roughing tables and the fines, after being dewatered, to three Diester slimers. The concentrate is stored for shipment.

Two Pelton-Doble wheels under a 250-foot head supply approximately 80 hp. An air lift is used to elevate the water from the sump to the drainage level, the air for this and the machines being furnished by a Denver hydrocompressor, which has a capacity of 400 cubic feet of free air a minute under a 250-foot head. In periods of low water, two Ingersoll-Rand two-stage Imperial type, steam-driven compressors are used. Sullivan, Ingersoll-Rand, and Excelsior Drill Co. machines are used underground.

READY BULLION MINING CO.

The Ready Bullion group of claims is situated about $1\frac{1}{2}$ miles from Hollis, on Twelve Mile Arm, an inlet of Kasaan Bay, on the east coast of Prince of Wales Island.

A horse tram with rails of 4 by 4's covered with strap iron has been built from the beach to the mine and is used for hauling supplies.

The well-defined vein, which shows several inches of gouge on both walls, is a few inches to several feet thick, averaging 6 or 7 inches. It strikes N. 25° W. to N. 35° W., and dips 35° to 50° NE. The country rock is altered tuff, slate, and quartzite. The property is opened by four drifts, the third or mill level having been driven for a distance of nearly 1,200 feet.

The method of mining is worthy of especial mention, as it has greatly reduced working costs. After a drift has been driven the hanging wall is shot down about 6 feet above the drift for a width of 3 feet and the ore is stripped. Heavy stulls are then set across the top of the drift and lagged. Every 50 feet a close-lagged oreway and manway is started, and between these three waste chutes are built. Another cut is then shot off the hanging wall; the ore is stripped on canvas and shoveled into the oreways. As the stopes proceed, just enough waste is drawn through the waste chutes to give headroom. The manways and oreways are kept tightly lagged, but the waste chutes are always built only a trifle above the stulls on the drift level, so only a small percentage of the waste is shoveled.

At the mill a 4-foot Pelton wheel, under a 347-foot head, furnishes power for the compressor and mill equipment, except the table which is driven by a 16-inch wheel. The ore passes over a $2\frac{1}{2}$ -inch grizzly to an 8 by 10 inch Blake crusher and then goes to the bins. From these it is fed to a Risdon 5-stamp battery with 60-mesh screens by a Risdon automatic feeder. The stamps weigh 1,000 pounds each and drop 7 inches 90 times a minute. The pulp passes over amalgamating plates to a Standard table, the concentrates being shipped to the Tacoma smelter.

RUSH & BROWN MINE.

The mine owned by Rush & Brown is on Prince of Wales Island, near the head of Kasaan Bay, on the northern side.

The ore bodies are of two types—magnetite-chalcopryite ore in a contact between intrusive diorite and altered sediments and chalcopryite in a shear zone in altered sediments. The former is known as the magnetite and the latter as the sulphide ore body. Both have been opened from a 250-foot shaft with three levels, and a winze is now being sunk to 300 feet. The ore from the magnetite body has been mined by a number of stopes and an open pit more than

100 feet deep; the workings in the sulphide body are stopes. Both ore bodies have a general east strike and dip south. Only shipping ore has been mined from either ore body, although large bodies of milling ore have been opened. The ore is washed and hand sorted in a sorting station and lowered on a balanced tram 300 yards to the mine bunkers. From here it is hauled 3 miles to the wharf and stored in bunkers, having a capacity of about 1,000 tons, for shipment to the smelter. During the past year the shipments have been to the Granby smelter at Anyox, British Columbia.

SEA LEVEL DEVELOPMENT CO.

The company, which had an option on the Sea Level and Sea Breeze claims the past season, pumped out the 100-foot shaft on the Sea Level, and extended the drift at the bottom 50 feet. About 70 feet of drifting was done on the Sea Breeze claim.

THE B. L. THANE EXPLORATION CO.

The company had an option on the Poorman group, about 2 miles northwest of Kasaan, on the east coast of Prince of Wales Island, and did a little development.

The ore is magnetite associated with a small amount of chalcopyrite.

SITKA DISTRICT.

CHICAGOFF MINING CO.

The high-grade gold bearing quartz vein in the Chicagoff mine lies in a shear zone in graywacke. The property, which is situated on Klag Bay, on the west coast of Chichagof Island, about 50 miles northerly from Sitka, and embraces the original Chicagoff mine and the adjoining Golden Gate ground, has been opened by a drift from which shafts 840 and 2,500 feet from the tunnel mouth have been sunk 800 feet on the ore. A 634-foot raise has been driven from this drift to the Golden Gate workings to provide an oreway and ventilation. Eight levels have been opened from the first incline and six from the second, and at the face of the main drift there is approximately 2,200 feet of backs.

There are two mills on the ground, the Chicagoff and the Golden Gate. In the former, which contains 20 stamps, the ore passes from the batteries over plates to a tube mill from which it goes to a second set of plates. From these the pulp passes over Diester tables and is finally treated by flotation. In the Golden Gate mill there are 10 stamps and the battery pulp after passing over amalgamation plates is treated on Wilfley tables.

PACIFIC COAST GYPSUM CO.

At the Gypsum mine, at Gypsum, on the east coast of Chichagof Island several bodies of gypsum have been opened, the largest being developed through a 185-foot 3-compartment shaft with levels at the 75 and 196 foot points. The deposit strikes southeast, dips about 30° northeast and is 100 to 150 feet wide. Drifts and crosscuts are run from the shaft and raises are driven from which stopes are opened on the full-breast system.

A Hendrie & Bolthoff twin-cylinder 8 by 12 inch double-drum hoist raises the gypsum to the mine bunkers, which have a capacity of 3,000 tons. From the mine the gypsum goes over a 1-mile railroad to the wharf, where it is stored in bins having a capacity of 2,000 tons. From there it is shipped to the company's plant in Tacoma, Wash.

SOUTHWESTERN ALASKA.

COPPER RIVER DISTRICT.

ALASKA CONSOLIDATED COPPER CO.

Development work was continued at the company's property by the eastern interests that hold an option. The mine is situated on Nugget Creek, a tributary of the Kuskulana, and is about 16 miles from Strela, a station on the Copper River & North Western Railroad. During the winter a power plant and compressor were installed.

GREAT NORTHERN DEVELOPMENT CO.

Only assessment work was done at the company's property, which is on Clear Creek, a tributary of the Kuskulana.

HUBBARD-ELLIOTT COPPER CO.

Development was continued at the company's property on Elliott Creek, a tributary of the Kotsina.

KENNECOTT COPPER CORPORATION.

The corporation is working two mines, the Bonanza and the Jumbo, on Bonanza Mountain, and is developing several prospects in the Copper River district. The mines are close together and about 3 miles from the concentrator at the terminal of the Copper River & North Western Railroad.

Both mines are connected with the concentrator by Bleichert aerial tramways and at both most of the ore is chalcocite with smaller amounts of covellite and copper carbonates. The ore occurs in irregular bodies and seams throughout the limestone; there is no regular method of mining, but a separate plan is adopted for each ore body, according to its size and slope. In the larger masses the

high-grade ore has been worked in benches; the mill-ore has been blasted in from the sides to fill the slopes and then drawn as needed. The limestone is hard and stands well, permitting the opening of chambers of a size that in a softer rock would be attended with considerable danger.

The Jumbo mine is opened by an inclined shaft, 30°, with levels 100 feet apart vertically. This shaft is being sunk from the 700 to the 800 foot level. Ventilation is natural except for the air from the machines and the small blowers used to force air into the dead ends of the drifts. The tramway from the mine to the concentrator, completed last year, has a capacity of 20 tons an hour at a speed of 528 feet per minute. It is 15,250 feet long and has 19 towers, three breakovers, and three tension stations. The track cable on the loaded side is 1½ inches in diameter, on the empty side 1 inch, and the haulage cable is seven-eighths inch. Each bucket has a capacity of 6 cubic feet.

The air-compressor plant at the Jumbo comprises two Ingersoll-Rand Imperial type, Class XB-2, 15 and 9 inches by 12 inches, cross-compound, 2-stage compressors; each having a piston displacement of 513 cubic feet of free air a minute belted to 79½-hp. Westinghouse type C. W. motors; one Ingersoll-Rand Imperial, class NE-1, 12½ by 12 inch single-stage compressor with herringbone gear connection to a 50-hp. Westinghouse type C. S. induction motor; and two small compressors, not in use at present time, each with a piston displacement of 310 cubic feet a minute.

The Bonanza mine is opened by a crosscut and inclined shaft sloping 26 to 33 degrees, and a glory hole. As the shaft is too small for good work and a larger part of it is in ore, a new 3-compartment shaft which will provide two hoisting compartments and a manway, is being raised from the 600-foot level to the main crosscut. This year a crosscut was completed on the third level to the Mother Lode side of the mountain and a raise driven to surface for air and a manway. All ventilation is natural except for the air from machines and that from small blowers for the dead ends of advance workings.

The Bonanza tramway, 15,450 feet long, with 44 towers, an angle station, and a tension station, has a capacity of 18 tons an hour at a speed of 452 feet a minute. The diameter of the track cable on the loaded side is 1½ inches, on the empty side seven-eighths inch, of the haulage cable eleven-sixteenths inch. The buckets have a capacity of 5 cubic feet each.

The air compressor is an Ingersoll-Rand, class PE-2, 25½ and 14½ inches by 21 inches, cross-compound, two-stage; with a piston displacement of 2,030 cubic feet of free air per minute, it is directly connected to a 250-hp. Westinghouse synchronous motor.

In the power plant, which is on the concentrator site, are two 305-hp. special Erie City oil-burning, water-tube boilers, which furnish steam at 160 pounds, working pressure. The prime movers are: An Allis-Chalmers 33 and 17 inch by 21 inch reciprocating engine, direct-connected to a 400 k. v. a. Allis-Chalmers class A-1 generator; a 500 k. v. a. Westinghouse turbo-generator and a Pelton wheel, direct connected to a 200 k. v. a. Westinghouse type G generator. Current is generated at 480 volts and used at this pressure in the mill; for transmission to the mines it is stepped up to 10,000 volts but is stepped down to 480 where it is used. Two oil tanks with a total capacity of 153,000 gallons hold the fuel oil.

At the concentrator the ore is crushed and passed over an automatic weighing belt, then it is sampled and hand-sorted. The high-grade ore is shipped without further treatment but the low grade is crushed and classified to obtain a high-grade concentrate, which is shipped, and a tailing which goes to the leaching plant.

A 6,095-foot tram, Leschen & Sons, connects the mine with 1,500-ton ore bins on McCarthy Creek, the difference in elevation of the terminals being 2,798 feet. Its capacity is 100 tons in an 8-hour day. The diameters of the cables are: Loaded side, 1½ inches; empty side, seven-eighths inch; and haulage, three-fourths inch.

The power plant consists of an 80-hp. twin-cylinder Otto gas engine, a 125-kw. 3-phase Payne generator, a 3½-hp. gas engine, which compresses air for the starting device of the Otto engine, a Sullivan, type W. C. 4, air compressor, driven by a 75-hp. motor, a direct current 7-hp. generator, connected with a 2½-hp. fan, a 7-hp. alternating current motor, to run a 7-hp. fan, a No. 5 Cameron sinking pump, worked by air or steam, and a double-cylinder 6 by 8 inch air or steam hoist. Sullivan drills are used underground.

MOTHER LODE COPPER MINES CO.

The company's property is about 1½ miles from the Kennecott Bonanza mine and 14 miles from McCarthy, a station on the Copper River & North Western Railroad. During the winter months the ore is sledged over a wagon road down McCarthy Creek to the station.

The ore body is 2 to 19 feet wide. The predominant mineral is chalcocite, which is associated with sulphides and carbonates. The country rock is limestone. The main ore body strikes generally N. 22° E.; it dips 80° E. in the northern part of the workings, and 70° W. in the southern part.

The property has been opened on four levels for a vertical distance of over 250 feet, the lowest level being tapped by a crosscut at an elevation of 5,585 feet. A 2-compartment shaft has been started on this level, though work has been temporarily discontinued on account of the high cost of power.

NORTH MIDAS COPPER CO.

The property is situated about 11 miles from Strelina, on the south side of the Kuskulana River. In the group are 12 lode claims covering a number of outcrops of copper-bearing ore, the predominant mineral being chalcopyrite. Four crosscuts have been driven to intersect the lodes.

PLACER MINING.

Most of the placer mining in the Copper River district was on tributaries of the Chistochina and Nazina Rivers, though a number of smaller properties were worked on the Brenner and other tributaries.

On Dan Creek, a tributary of the Nazina, the Dan Creek Mining Co. works a group of claims. As the creek has a comparatively rapid fall and is extremely turbulent during high water, this company has had to meet many difficulties. In opening the ground the main body of the creek is flumed past the area to be worked, wing dams set, a bedrock flume laid, and the gravel washed into the flume by giants. Railroad iron is used for riffles and a tailing giant at the lower end of the flume stacks the waste. As the pits are worked out, the wing dams are moved up stream and the flumes extended.

On Chititu, also a tributary of the Nazina, Esterly & Andrus had the largest placer operations, working one plant on the main stream and a second on Rex, one of the principal branches. On Chititu, a bedrock flume is laid in the spring through the part of the ground to be worked that season. Giants are set at the upper end of the block and the pit worked downstream, the sluices being extended as necessary to carry the tailings. Railroad iron and sawed blocks are used for riffles.

Near the mouth of Rex, Frank Kernan worked a small hydraulic plant, and a few claims up the creek E. W. Brooks employed a few men on similar work. Bert Carvey worked a bench claim on the same creek when water was available.

On the Chitistone, Slate Creek and its tributary, Miller Gulch, were the center of activity, though scarcity of water on Miller Gulch prevented extensive work. At the mouth of Slate Creek J. M. Elmer and associates worked three placer claims under lease. A 20-inch pipe conveyed the water to one 4-inch and two 5-inch giants under a 140-foot head. The gravel bank was approximately 20 feet high.

Several claims above this is the John S. Miller group of three claims. Here water from two streams under a 280-foot and a 390-foot head is directed against a 20-foot bank by one 4-inch and one 3-inch nozzle. Meyer & Prolig, Kraemer, Leavell & Hemple, William Schroeder, Bert McDowell, and the Canaille brothers all worked small plants in the same locality, and a number of others did assessment work on their claims. The gravels of the lower Chesna were drilled by John Haze-

lett to test their availability as dredging ground. On the upper Chesna, Holmes & Brail worked a small plant.

KENAI PENINSULA.

ALASKA CROW CREEK MINING CO.

The company does hydraulic mining on a group of 29 creek and bench claims on Crow Creek, a tributary of Glacier Creek, about 5 miles above its mouth. Glacier Creek flows into Turnagain Arm from the northern side, crossing the surveyed line of the Government railroad at Glacier. The group covers several miles of the creek bed.

Much of the company's energy in the past has been expended in efforts to get the property in good working condition and the work was not completed until the fall of 1915. The gravels toward the lower end of the creek, most of which were comparatively barren, were sluiced to bedrock and a flume built to pass the stream around that point. A bedrock flume 2,200 feet long, 5 feet wide and 5 feet deep was built in this cut on a grade of 7 inches to the box length (12 feet). At the intake the floor of the first box is made of manganese steel plates. The next seven have railroad iron for riffles and the rest have 14-inch sawed, wooden blocks.

A ditch about a mile long conveys the water from the upper part of Crow Creek to the penstock which gives a head of 300 to 400 feet at the bottom of the pit. The main pipe line is 24 inches in diameter, several giants being operated, according to the stage of the water.

The property is equipped with a sawmill that furnishes lumber for the flume and dwellings.

ALASKA SECURITIES CORPORATION.

The corporation has recently consolidated several groups of placer claims on Lynx Creek, where it is installing a plant, and is drilling another group on Canyon Creek, just above the forks of the Government road.

On the Lynx Creek group a ditch nearly a mile long and a 4,000-foot pipe line is under construction which will give a 400-foot head. A tunnel has been driven through a low ridge to give room for the disposal of the tailing, and a bedrock flume has been laid. Hungarian riffles are used for the first two box lengths, and sawed blocks for the remainder.

On Canyon Creek the company has had a Keystone drill busy testing an old channel the past summer.

BLUEBELL AND PRIMROSE MINES.

Some development work was done at the Bluebell and Primrose groups about five miles southwest of mile post 18 on the Alaska Northern Railroad. Several veins on both groups have been pros-

pected superficially by short drifts and crosscuts and at the Primrose a small Little Giant mill has been installed. The veins, quartz in a slate-graywacke country rock, carry gold. Most of the ore is free milling. Shoots of arsenopyrite contain considerable gold and silver and during the past season a test sample of 10 tons of this ore was shipped to the Tacoma smelter. Both groups have been under the same management for the past two years.

GILPATRICK MINE.

Only assessment work was done at the claims situated about 15 miles from mile post 29 on the Alaska Northern Railroad during the past season. There are several veins, but the greatest amount of work has been done on a mineralized dike adjacent to one of them. This dike is well above timber line and has been opened by several hundred feet of drifts and raises. An arrastra has been installed on the creek near the bottom of the canyon to treat the soft oxidized ore from the surface. The high-grade ore is sacked and hauled down the mountain on sleds.

HERRON DREDGING CO.

The dredge near Sunrise was idle during the past season.

KENAI-ALASKA GOLD CO.

The company owns a group of five lode and three placer claims on the north side of Falls Creek, about 4 miles from mile post 26 on the Alaska Northern Railroad. The mine, which is situated at an elevation of 4,500 feet, has been opened by two crosscuts with connecting raises and stopes. It is connected with the 5-stamp Hendy mill by an 8,200-foot tram which has a fall of 2,400 feet between terminals. This property was under lease a part of the year to Drenan & Sweitzer.

LUCKY STRIKE MINE.

The Lucky Strike group of five lode claims is owned by John Hirshy; it is situated on a ridge at an elevation of 3,500 feet, at the head of Palmer Creek, a tributary of Resurrection Creek, about 14 miles from Hope.

The lode, a series of veinlets in a crushed and broken slate, strikes east and has a variable dip to the northwest. Two crosscuts have been driven to intersect the ore, but owing to limited milling facilities only the high-grade stringers have been worked. A 1,000-foot jig-back tram takes the ore to a flat whence it is hauled about half a mile on sleds to the mill. The ore passes through an 8 by 10 inch jaw crusher and drops into a 10-ton bin, whence it is fed by a Challenge feeder to a Moyle one-stamp battery, using a 30-mesh screen.

The stamp weighs 400 pounds, has a 6-inch drop, and falls 110 times a minute. The pulp passes over amalgamating plates to an Ogden concentrator and is stored for future treatment. The concentrate is shipped.

MATHISON MINING CO.

The company is working a group of 18 claims and a fraction on Resurrection Creek, about $3\frac{1}{2}$ miles from Hope. The main flow of the creek is carried along one bank, while the opposite side is worked in a series of pits, as the creek is wide and there is but little fall. A 48-foot sluice, 4 feet wide, with a grade of 7 inches to the box length, has small railroad iron for riffles. Three or four No. 2 giants are kept busy, according to the stage of the water, which is brought in under a 350-foot head. The tailing is washed clear from the sluices with a tailing giant. A steam shovel has been at work for a few seasons digging a 7-mile ditch for more water.

PEARSON BROS.

The Pearson Bros. worked a group of eight claims on Resurrection Creek, about 6 miles from Hope. The creek water is carried in a ditch along one bank, while the creek bed is worked in a series of pits. The gravel is 5 or 6 feet deep and comparatively free from large boulders. Water for hydraulicking is brought to a penstock, which gives a fall of 310 feet. At the intake the hydraulic pipe has a diameter of 24 inches, which is reduced to 11 inches where the water is distributed to the giants.

RONAN & JAMES.

Ronan & James have a lease and bond on the Champion and Gladiator lode claims in the Moose Pass district, where they installed a small arrastra this season.

The vein, about 12 inches wide, strikes N. 60° E. and dips southeast. A crosscut from a 50-foot shaft taps the ore 85 feet below the shaft collar. Drifts have been started along the vein at the crosscut and a stope is being driven to tap the shaft and furnish ventilation.

SCHNEEN-LECHNER MINE.

Only assessment work was done at the Scheen-Lechner group during the season. The group is on Falls Creek, about $4\frac{1}{2}$ miles from mile post 25 on the Alaska Northern Railroad.

ST. LOUIS MINING & TRADING CO.

The company is working a group of placer claims on Resurrection Creek, about 4 miles from Hope. The creek water is carried along one side of the channel while a series of pits on the other side are worked out. Giants wash the gravel into a 36-foot flume, $4\frac{1}{2}$ feet

wide, in which 60-pound railroad iron is used for riffles. Another giant stacks the tailing. A 30-inch pipe line, reduced gradually to 11 and 9 inches at the giants, brings the water $1\frac{1}{2}$ miles and gives a fall of 137 feet.

MATANUSKA-SUSITNA DISTRICT.

ALASKA FREE GOLD MINING CO.

The company's property, under lease to William Martin, of Seattle, is on Fishhook Creek, just to the west of the Independence.

Several rather irregular ore bodies have been opened at a number of different points without any special method of development, though connections have been made on the ore from the east to the west side of the mountain. The ore bodies are all in a blocky diorite, strike approximately N. 20° W., and dip 38° to 40° southwest.

The mine workings are about 1,000 feet above the mill, with which they are connected by three aerial tramways. On the east side of the mountain a 750-foot tram connects the mine bins with an intermediate bin, from which the ore is trammed over a track about 500 feet long to a second mine bin. From here it is lowered to the mill over a single-span, 2,100-foot tram. West of this a single-span, 2,400-foot tram, connects another set of mine bins with the mill, and farther west, a third tram, 4,500 feet long with two towers, connects another set of workings to the mill. All track cables have a diameter of $\frac{1}{4}$ inch and the haulage and tail ropes $\frac{1}{2}$ inch. The buckets have a capacity of approximately 450 pounds.

At the mill bins, which have a capacity of 80 tons, the ore is crushed by a jaw crusher to $1\frac{1}{2}$ inches and is fed automatically to two 10-foot slow-speed Lane mills making 6 to 8 revolutions a minute. The size of the pulp is governed by the height of discharge and the amount of feed water. Quicksilver is fed to the mills and the pulp is split into four parts, going over 5 by 10 foot plates set on a grade of $1\frac{1}{2}$ inches to the foot. From these it passes over two Barnes concentrators to classifiers and then to the cyanide plant. The slime is impounded for possible future treatment, and the concentrate is shipped to the smelter.

The cyanide plant comprises four 30-ton leaching tanks—two sump and two solution tanks—the pulp being given a 4-day treatment. The gold is precipitated on zinc shavings; the precipitate is dried, roasted, and shipped to the smelter.

Power for the plant is furnished by a 10-inch turbine under a 54-foot head, a 25-horsepower and a 16-horsepower gasoline engine all belted to the same shafting.

CACHE CREEK DREDGING CO.

The dredge erected on Cache Creek during the past spring represents the first serious attempt at gold dredging in this part of the Territory.

The bedrock includes slates, graywackes, sandstones, and shales, with occasional beds of lignite. The creek is comparatively shallow, its average depth being approximately 6 feet, so the dredge has a large hull, 87 feet long, 54 feet beam, and 7 feet deep, for shallow draft. The planks of the bottom are 4 by 12 inches, those on the sides 6 by 12 inches, and the deck 3 by 12 inches. The winch deck is 30 feet above the main deck and the pilot deck is 12 feet above this. The spuds, made of wood reinforced with steel, are 44 feet long and weigh 11 tons each.

The 65 close-connected 7-foot buckets have manganese-steel lips; and although the dredge has a capacity of 3,000 cubic yards in 24 hours, the management has based its plans on an output of 2,000 cubic yards, on account of the shallow ground and the number of bowlders. The latter have retarded the speed of the dredge, but have not proved a serious hindrance, as they are not large enough to require special devices for handling. With the present number of buckets, the dredge can dig 30 feet below water line, though it probably will not be called upon to work at anywhere near this distance on this part of the creek.

The gravel from the buckets falls over an 11-inch grizzly into a 48-inch flume, 108 feet long. The grade of the flume is adjustable, though the best results are obtained by keeping it at a fall of approximately $1\frac{1}{2}$ inches to the foot. The riffles are 2 by 4's capped with $\frac{3}{4}$ -inch manganese-steel plates and are set $1\frac{1}{2}$ inches apart. A 6-inch centrifugal pump supplies wash water at the grizzly and an 18-inch centrifugal pumps directly to the flume. The buckets are washed clean by water from two nozzles and the washings fall to a save-all sluice with Hungarian riffles. This sluice is 18 inches wide and set on a grade of 1 inch to the foot.

A feature never used before on an Alaskan dredge is the construction of rock chutes from the grizzly. The oversize from the bars passes to a Y. The branches of the Y pass on each side of the save-all sluice and empty in the pond aft of the dredge, where the rock forms a dam and prevents the fines from filling the pond under the boat. These chutes are 42 inches wide and lined with manganese-steel plates at points of greatest wear.

Power is furnished by a 250-horsepower Yarrow tubular boiler, using coal mined locally for fuel. A 150-horsepower Reaves engine drives the pumps and dynamo, a 125-horsepower Lidgerwood the digging ladder, and a 20-horsepower the winches. The exhaust steam from the engines passes to a condenser.

A number of beds of lignite outcrop on Cache Creek and its tributaries from which an ample supply of fuel may be had at low cost. The company has obtained a free-use 10-acre permit to mine coal and in the summer of 1916 obtained its supply from an open cut about 2 miles from the dredge, but a contract has been let to drive entries and open rooms on a 5-foot bed on the creek during the winter, so that fuel will be easily available at a convenient point in the spring. The coal burns with a long flame almost like wood, and makes a good steaming coal, though the percentage of ash is high. If the company had not owned the steam equipment before the boat was planned, the situation would have been ideal for electric operation by the power generated at the point where the coal was mined.

After a short though successful season, the only change planned on the dredge is a second bull wheel, so that the drive will be on both sides of the bucket line.

DOHERTY COAL MINE.

In the spring of 1916 a contract was let by the Alaskan Engineering Commission to R. C. Doherty to furnish 2,000 tons for the use of the railroad, and Mr. Doherty opened a small mine on the west bank of Moose Creek, about a mile above the railroad right of way.

An 800-foot drift was driven on a bed which strikes S. 50° W. to S. 70° W. and dips 40° to 53° SE. Fifteen feet above this an airway was driven and rooms were turned from it. The 100-foot rooms were 20 feet wide with 10-foot pillars, but later rooms were driven double, or 50 feet wide, with 30-foot pillars. They were driven along the bed to the conglomerate that covers the outcrop except in the creek valley. The coal is lignitic, and the bed gives off a little gas, although ventilation is entirely natural, except a small homemade fan in the dead ends has been used. The thickness of the bed varies from point to point, averaging more than 3½ feet. A typical cross section is as follows:

Section of lignite bed on Cache Creek.

Material.	Thickness.	
	Ft.	in.
Hard sandstone roof.		
Scaly sandstone with stringers of coal.....	0	4 to 6
Hard black coal with many "niggerheads".....	1	6
Shale parting.		
Fairly hard black coal with irregular fracture and no well-developed cleat.....	1	0
Lignite coal with woody structure.....	1	6
Soft dark brown shale.....	0	6
Hard sandstone bottom.		

Several other partings in the bed lack regularity. In places the sandstone footwall is replaced by fire clay.

The coal is hauled in cars, capacity 4 cubic yards, from the mine to the railroad bunkers by a 35-horsepower locomotive.

INDEPENDENCE GOLD MINES CO.

The company is working the Independence, a lode gold mine, near the head of Fishhook Creek, a tributary of Little Susitna River.

There are two veins, the Granite Mountain and the Independence, in a blocky diorite. The strike of both veins is approximately N. 10° W.; the Independence dips about 40° and the Granite Mountain 0° to 26° SW. The workings on the Independence are at an elevation of 4,650 feet; those on the Granite Mountain 4,100; and the mill is at an elevation of 3,650 feet. On the Granite Mountain, where the bulk of the work has been done, a drift has been run 529 feet on the vein. From the end of the drift an incline, which flattens about one-third of the way down, has been sunk 480 feet on the vein and from these two both overhand and underhand stopes have been driven wherever the value of the ore warranted. Stulls are used to support the roof, and the waste is used for pack walls.

The bins on the Independence are connected with the mill by a single-span, 2,800-foot aerial tramway with $\frac{3}{4}$ -inch track and $\frac{1}{2}$ -inch haulage and tailrope cables. The Granite Mountain bins are connected to the mill by a 1,730-foot tram with similar cables. Both tramways carry 400-pound buckets.

At the mill the ore passes over a 1-inch grizzly to the bins, the oversize going to a 7 by 9 inch Dodge crusher. From the bins it is fed automatically to a 3-stamp battery of 500-pound stamps and one 1,300-pound Nissen stamp. The practice is to crush to 40 mesh, using both inside and outside amalgamation; the pulp from the batteries flows over plates set on a grade of 1 $\frac{1}{4}$ inches to the foot. From the plates the pulp passes over an Ogden concentrator to tailing ponds. The mill capacity will be increased this year by a 5-foot, 6-inch Denver mill, with a rated capacity of 30 to 40 tons in 24 hours when run at a speed of 30 revolutions per minute. The pulp from this mill will pass over amalgamating plates to a Wilfley table and be stored for future treatment.

MABLE MILLING, MINING & POWER CO.

The company is opening a lode mine on Archangel Creek, a tributary of the Little Susitna, just above Fishhook. Some open cuts have been made and a drift run on the ore, which is in a blocky diorite. A 15-ton Denver mill is being installed about 1,200 feet below the mine and a 3-500-foot tramway with one tower is being erected to convey the ore from the mine to the crushing plant.

WILLOW CREEK MINES CO.

The company is working the Gold Bullion mine, on the divide between Willow and Craigie Creeks, under lease and bond, and is also operating the Nugget property, which joins the Gold Bullion on the

Willow Creek side and the Brooklyn Development Co.'s ground on the west side of Willow Creek.

The mine is situated at an elevation of 4,500 feet and, as the vein is comparatively flat, has been opened by a number of drifts. These have been started where the value of the ore on the surface warranted and an irregular system of overhand stoping followed, according to the size and value of the ore bodies. The roof is supported by stulls and pack walls built with the waste. The vein, which fills a fissure in a grano-diorite, varies from 8 inches to 7 feet thick, has an average strike of S. 10° E., and dips approximately 14° NW. Through the diorite, which is rather blocky, are numerous quartz stringers, some of which are large enough and contain enough gold to be profitable mining.

Owing to the nature of the ore bodies, the mine bins are connected by a light aerial tramway, which can be easily moved, with intermediate bins, and there are connected to the mill by an aerial tramway of more substantial construction. The mine bins are 700 feet above the intermediate bins and the latter 800 feet above the mill. One 2,500-foot span connects the mine and intermediate bins with three-fourth-inch track and one-fourth-inch haulage and tail-rope cables which carry two 5-cubic-foot buckets. This tram runs entirely by gravity. The line from the intermediate bins to the mill is 3,800 feet in length with seven towers. The track cables are seven-eighths of an inch in diameter, the haulage and tail rope one-fourth inch, and the buckets contain 7 cubic feet. For a short distance of the bucket travel it is necessary to use a small amount of power. As the mine is far above the timber line it is necessary to send up all mine timbers as well as supplies for the mine camp over these tramways.

At the mill the ore goes over a 7 by 9 inch Blake crusher to bins having a working capacity of 160 tons, from which it is fed to the stamps by Challenge feeders. There are three batteries, two 5-stamp, and one 2-stamp, which crush the ore to one-half millimeter, using a diagonal slot screen. Seven of the stamps weigh 1,050 pounds each, the others 850 pounds, but these will probably be increased in weight at a later date. They fall 6 inches and drop 105 times per minute. Both inside and outside amalgamation is used. From the batteries the pulp flows over standard 5 by 10 foot plates set on a grade of 1½ inches to the foot. The pulp is carried by launders to a hydraulic classifier, the oversize going to a second classifier, the spigot product to a Wilfley table. The overflow from the second classifier is sent to the slime pond for storage and the spigot discharge to a second Wilfley. The Wilfley concentrate is treated on the ground or shipped to the Selby smelter, according to its value. The sand is pumped to the cyanide plant, which consists of five 30-ton leaching tanks, two

sump tanks, one make-up and one storage tank. The pulp is given a four-day treatment and the gold precipitated on zinc shavings. The precipitate is dried, roasted, and shipped to the Selby smelter.

The power for the mill is furnished by a 12-inch reaction turbine under a 25-foot head with a 12-foot draft tube. A 2,000-foot pipe line, consisting of four sections 500 feet each of 8, 7, 6, and 5 inch pipe, is being laid to furnish additional power.

This spring the mine, intermediate, and mill bins have been enlarged and the five 850-pound stamps added to the equipment, giving the plant a capacity of approximately 40 tons per 24 hours.

PLACER MINING.

Placer prospects may be found scattered over a large part of this district, but the most productive localities are on Cache and Valdez Creeks.

On Cache Creek the largest operator is the Cache Creek Dredging Co., previously described. The Thunder Creek Mining Co. operated three plants—one on Falls, one on Windy, and one on Thunder Creeks, all tributaries of Cache.

On Thunder, a comparatively short creek, work was carried forward near the lower end and center of the claims. The gravel, being shallow (2 to 4 feet deep), was sluiced off in a series of pits in each section with 3-inch nozzles. The method of operation was to set the sluices, which were 20 inches wide, on a grade of 5 inches to 12 feet in a worked-out pit. A small wing dam was built at the upper end of the sluice and a pit 100 to 150 feet long and about 40 feet wide worked out. The sluices were then moved to the opposite side of the creek and the gravel there washed off. The boxes were then moved forward and the bedrock of one of the first pits shoveled while the cycle was repeated on the gravel above.

On Windy Creek the Thunder Creek Mining Co. operated two 4-inch giants under a 225-foot head against a 225-foot bank of gravel.

The other operators of this locality were Tesmer & Beidermann and William Peterson, on Cache Creek; Bubb & Bahern, on Dollar Creek; Ebheart & Anderson, McElroy & Remmer, J. C. Funk, and A. A. Adams, on Falls Creek; Gage & Mack, on Thunder Creek; Harper Bros., Smith & Hogan, Hugh Price, and Carl Raymond, on Nugget Creek; Weatherall & Andresen, on Gold Creek; Chris Hamersmith and William Gredaken, on Bird Creek; Frank Jenkins, John Rice, and Chris Hansen, on Willow Creek; Kast, Nelson & Larson, on Poorman Creek; Richard Richardson, on Ramsdyke Creek; Francis & Foster, on Long Creek; Van Eiderstein, on Peters Creek; Wolf & Maloche, on Spruce Creek; and John Gray, on Treasure Creek.

On Valdez Creek the largest operations were carried on by the Valdez Creek Placer Mines Co., which has been installing a hydraulic

plant and preparing for work on a large scale for the past few years. Clark Duff and C. J. Carlson operated in a small way in the same district. A number of others did assessment work on their holdings.

PRINCE WILLIAM SOUND.

ALASKA MINES CORPORATION.

The corporation operates the Schlosser property, about three-fourths of a mile from the beach, on the south side of Fidalgo Bay. The ore, chalcopyrite mixed to a greater or less extent with massive iron sulphide, is found in lenses along a shear zone which vary in size but have a fairly consistent strike of N. 20° E. and are nearly vertical.

The property has been opened by drifts on four levels, the third or tram level being at an elevation of 796 feet. The fourth level is 138 feet below the third, the second is 129 feet above it, and the first 75 feet above the second. More or less stoping has been done on all the levels and connections have been made between the third and second.

A 600-ton ore bunker has been erected at the wharf and connected to the mine by an aerial tram. The diameters of the cables are five-eighths, seven-eighths, and 1½ inches, respectively, for the traction, empty track, and loaded track cables.

ALICE MINES (LTD.).

The property of the Alice Mines (Ltd.) is situated on the north shore of Shoup Bay. The fissure crosses a slate-graywacke series with a strike of N. 62° W. and a dip of 20° to 80° S. The vein is 6 inches to 2½ feet wide. It has been opened by a 250-foot drift and a 170-foot shaft, the bottom of which is 100 feet below the drift level. From the bottom of the shaft drifts have been driven 50 feet in each direction along the vein.

The property is equipped with an 8 by 10 inch twin-cylinder steam hoist, 100-horsepower boiler, a compressor with a capacity of 360 cubic feet of free air per minute, machine drills, pumps, and electric lighting plant.

BANNER MINE.

The Banner group of claims on the south shore of Bettles Bay, a small body of water off Port Wells, has been developed by C. Christopher, under bond, during the year. The vein is a fracture zone in a siliceous dike, which strikes northeast and dips steeply northwest, the broken material being cemented with gold-bearing quartz. A drift has been driven along the ore, which varies from 1 to 6 feet wide, for about 200 feet. It is planned to install a mill and mining machinery in the spring.

CLIFF MINE.

At the Cliff mine, on the north shore of Port Valdez, about 12 miles from the town of Valdez, two drifts were driven above the old workings and a raise started to connect them. The ore from this development was hauled to the mill and crushed whenever a sufficient amount had accumulated to warrant starting the plant, which contains six Nissen stamps.

CUBE MINES CO.

The company is developing the Three-in-One and Cube groups of claims on the north side of Port Valdez.

On the Three-in One the vein varies from 2 to 8 feet wide and crosses a country rock of graywacke. At an elevation of 1,450 feet a drift has been run for 900 feet in the vein and 175 feet above this a drift has been driven 400 feet. The two have been connected by a raise. The company has purchased equipment consisting of a Buchanan jaw crusher, a Hardinge conical mill, three Faust tables, and two Fairbanks-Morse 25-hp. semi-Deisel engines, which will be installed at the property this winter.

At the Cube a drift has been driven 30 feet on complex ore containing copper, iron, and gold. It is planned to ship this material to the Granby smelter at Anyox, B. C.

ELLAMAR MINING CO.

The company operates the Ellamar mine, at Ellamar, on the eastern shore of Virgin Bay, about 20 miles southwest of Valdez. The mine is opened on seven levels from a 600-foot, 3-compartment, vertical shaft, crosscuts being driven from the shaft to the ore body. The outcrop is covered with water at high tide, so a cofferdam has been constructed to prevent flooding. The ore body, a fracture zone in a group of sedimentary rocks, chiefly slates, has a maximum width of 100 feet, and is divided longitudinally into two classes of ore by a layer of slate which has an average width of 4 feet. The northeast or hanging-wall side is heavy, fine-grained pyritic ore with a low copper content and a fair value in gold. On the foot-wall side of the slate the ore contains a higher percentage of chalcopyrite than on the hanging-wall side but its gold content is lower. In the early years of operation, when only the richer copper ore was handled at a profit, the ore was mined through a glory hole and stopes worked on the shrinkage system. For a few years after the old stopes had been emptied no particular method of mining was employed but in 1913 a filling system was adopted whereby the old openings were cribbed and filled and the adjoining ore mined.

For the first few years the waste for filling was obtained from crosscuts and raises in the hanging wall. Owing to the high cost of the

filling and to the fact that there was considerable methane in the graphitic slate of the hanging wall, which accumulated in the raises and resulted in several accidents, the filling system was changed. The filling is now obtained from a glory hole, 700 feet from the ore body. It is drawn through chutes to the 100-foot level and hauled by a gasoline motor to the mine, where it is distributed by waste chutes to the particular workings where it is needed. Cribbed chutes are carried up and the stopes filled as the ore is mined. The ore is trammed by hand to the shaft, hoisted, and stored in bins from which it is loaded by an aerial tramway directly to the ship's hold. This year the work has been confined to those parts of the mine above the third level and to diamond drilling.

In the main power plant there is a 250-hp. Geary water-tube boiler and two 70-hp. Brownell locomotive-type boilers. The latter are used as auxiliaries in case the Geary is closed down. Crude oil is used for fuel, the storage tank having a capacity of 1,500 barrels. The hoist is a 10 by 12 inch, duplex-g geared, double-loose drum, Vulcan hoist. The compressor is a Nordberg two-stage, cross-compound condensing machine with 12 and 24 inch steam cylinders. The air cylinders are 12½ and 21 inches in diameter and the stroke is 36 inches. Ingersoll-Rand jack hammers, stope hammers, and type C-110 machines are used underground.

The tramway is of Brodrick & Bascom design. The capacity of the buckets is 14 cubic feet each and the capacity of the tram is 200 tons per hour with a rope speed of 450 feet per minute. The diameter of the track cable on the loaded side is 1½ inches, on the empty side 1 inch, and that of the traction cable is five-eighths of an inch.

FIDALGO MINING CO.

The company is operating a group of 24 lode claims on the southeastern shore of Fidalgo Bay, about one-half a mile from the beach.

The ore, chiefly chalcopryite, lies in a shear zone through slate, graywacke, and greenstone, and strikes N. 30° W., with a dip of 67° NE. Two drifts have been driven along the shear zone to open the ore with raises through to the surface. The ore occurs in lenses of variable size through this zone, the largest so far opened being 6 feet wide. The elevation of the upper drift is 962 feet; the lower one is 100 feet below this. About 500 feet of track has been laid from the lower level to the ore bins, which are connected with the bunkers on the wharf by a 2,000-foot Riblet balanced tramway. The buckets have a capacity of 10 cubic feet each and there are three towers. The track cables are 1 inch in diameter and the haulage cable three-eighths of an inch. The upper bins have a capacity of 50 tons and the lower 500 tons. The ore is hand sorted and shipped to the Tacoma smelter. This year a power plant has been installed containing a

36-inch Pelton wheel belted to an Ingersoll-Sargent compressor having a capacity of 385 cubic feet of free air per minute. The water for the wheel is conveyed 1,200 feet in a wooden-stave pipe, through which a fall of 400 feet is obtained. The upper 600 feet of the line is built of 12-inch pipe; the lower, of 8-inch. Ingersoll-Rand jack hammers are used underground.

GALENA BAY MINING CO.

The assessment work was the only development reported at the Galena Bay mine, on the ridge between Galena and Boulder Bays.

GOLD KING MINE.

The mine near the head of one of the eastern arms of Columbia Glacier, about 22 miles from Valdez, was operated for a short time the past season by the original owners. There are several known veins on the property which cut a slate-graywacke series.

The mill equipment consists of a 6 by 8 inch Dodge crusher, a 3½-foot Huntington mill, and a Frue vanner.

GOLDEN EAGLE MINE.

Only the assessment work was done at the Golden Eagle group, near Golden, in the Port Walls district. The property, which reverted to the original owners sometime ago, is equipped with a five-stamp mill.

GRANBY CONSOLIDATED MINING, SMELTING & POWER CO. (LTD.).

The Granby company operates the Midas mine, in Solomon Basin, across the harbor from Valdez. The ore, principally chalcopryrite in a silicious and pyritic gangue, strikes nearly east and west, and dips approximately 45° north. The ore body, which varies from 1 to 20 feet wide, average about 4½ feet, has been opened by three drifts, with crosscuts and raises, giving over 200 feet of backs. Much of the ore is sufficiently high grade to ship direct to the smelter, so the stopes have been confined to these high-grade areas in the mine, although this work has opened large quantities of ore of an excellent milling grade. It is planned to install a concentration plant at some later date.

During the year a 200-horsepower Diesel engine has been installed which drives a two-stage Ingersoll-Rand Imperial type air compressor with a capacity of 1,000 cubic feet of free air per minute. Ingersoll-Rand, Waugh, and Sullivan machines are used underground.

A 5½-mile Riblet aerial tramway, with a capacity of 15 tons per hour, conveys the ore from the mine to the bunkers at the wharf. The track cables are 1½ inches in diameter on the loaded side and seven-eighths inch on the empty side with a five-eighths inch haulage

cable. The tram is driven by a 30-horsepower Foos gas engine. The bunkers on the wharf have a capacity of 3,000 tons and load directly on ocean-going vessels with a belt conveyor driven by a 25-horsepower gas engine. The ore is shipped to the company's smelter at Anyox, British Columbia, on Observatory Inlet. About 14,000 tons was shipped to Anyox during 1916.

GRANITE GOLD MINING CO.

The Granite mine is about a mile from the beach, at Hobo Bay, on Port Wells. The vein varies from $1\frac{1}{2}$ to 8 feet wide, averaging approximately 26 inches, has a variable strike, and dips roughly 45° N. The fissure cuts a slate-granite contact, so a part of the ore body is in slate and a part in granite. There are many branches and considerable difficulty has been met in following the ore.

The property was originally opened with a crosscut to an inclined shaft on the ore, since which time a second crosscut has been run on the mill level and a raise driven to tap the bottom of the shaft. The level was opened on the main vein at 50, 110, 150, 210, and 350 feet, the mill crosscut being 125 feet below the latter.

The mill is in two parts, ten stamps on one side and a 7-foot Lane mill on the other. The ore on the stamp side passes through a jaw crusher to the bins where it is fed automatically to two Hendy 5-stamp batteries, the stamps weighing 1,350 pounds each, falling 105 times per minute with a $6\frac{1}{4}$ -inch drop. The ore is crushed to 40-mesh passed over amalgamating plates, and concentrated on Wilfley and Deister tables. In the second unit the Lane is followed by 14-inch Allis-Chalmers rolls, the pulp going over plates to a Wilfley table. The concentrate from both units is shipped and the tailings stored.

The main power plant, situated on the beach, contains two 80-horsepower oil-burning boilers which furnish steam for an American Ball 180-horsepower engine. The latter drives a 160-kilowatt Westinghouse dynamo. The compressor at the mine has a capacity of 620 cubic feet of free air per minute and is driven by a 100-horsepower motor. Underground, Sullivan and Ingersoll-Rand machines are used.

KENNECOTT COPPER CORPORATION.

The Bonanza mine, known locally as the Beatson or Latouche property, is owned by the Kennecott Copper Corporation. The property is situated at Latouche on the western side of Latouche Island near the northern end. The ore is chalcopyrite in an altered slate and graywacke associated with considerable pyrrhotite. The mine has been opened by a main-haulage level, with crosscuts through the ore body; a 100-foot level below this, three bluff levels at varying intervals above, and a gravity hole. The latter is roughly 100 by 400

feet in extent and is connected by raises to the haulage level, as are the bluff levels above.

During the summer the bulk of the ore mined is blasted from the face of the bluff into the glory hole, where it is bulldozed and drawn through to the haulage level. When the weather becomes too inclement to work outside the ore is drawn from stopes underground, which are worked on the full-breast system, only enough ore being drawn from the stope to give headroom for the machines. The broken ore passes over heavy grizzlies through bulldozing chambers before it enters the chutes and is reduced to a convenient size before being drawn into the cars. Some of the ore is shipped direct to Tacoma but the larger portion goes to the concentration plant.

In the mill the ore passes from a rotary coarse crusher to a Symons disk machine from which it is distributed by a conveyor belt to the bins. From these it passes to a Marcy mill, where it is ground to 40 mesh. It is next classified, the oversize going to Hardinge mills after which it unites with the undersize and passes to the flotation units. Following these is a Dorr thickener and an Oliver filter press, the concentrate being dried in a Ruggles-Cole drier before it is shipped to the smelter at Tacoma. The plant has a capacity of 750 tons per 24 hours but this is to be doubled in the near future.

The steam plant consists of three 305-horsepower Erie City water-tube boilers, using crude oil for fuel, which is stored in one 5,000 and one 25,000 barrel tank. Two 625 Westinghouse-Parsons turbines supply electricity for the mill and mine pumps.

Several types of Ingersoll-Rand and Sullivan machines are used underground, the air being supplied by an Ingersoll-Rand two-stage compressor, which has a capacity of 720 cubic feet of free air per minute. The latter is driven by a 122-horsepower McEwen engine.

LANDLOCK BAY COPPER CO.

The company owns a group of seven lode claims on the south side of Landlock Bay. The four ore zones, chalcopyrite, occupy shear zones through slate, graywacke, and greenstone. On the west side of the ridge on which the claims are located, two crosscuts have intersected the ore, and shallow winzes are sunk on the ore bodies. A wharf and 800-ton bunkers have been constructed near the entrance to the lower crosscut, which is about 80 feet above the sea level.

MINERAL KING MINING CO.

The company has been installing a 10-stamp mill this season at its property about a mile east of Bettles Bay, on Port Wells. There are three gold-bearing veins on the property, but except for a few open cuts the exploration work has been confined to one. This vein, which cuts a slate-graywacke series, strikes N. 40° W. and

dips 53° NE. Its width varies from 6 inches to 4 feet, with an average of 18 inches. A shaft has been sunk 110 feet on the ore and drifts run on the vein about 200 feet from the 100-foot point. The property is equipped with a 16-horsepower boiler and a 12-horsepower hoist. The mine is at an elevation of 700 feet and a mill site has been staked on the flat near tidewater, where water is available for power.

RAMSEY-RUTHERFORD MINING CO.

The company is operating a high-grade lode gold property, about 11 miles northeast of Valdez. The property is situated on a ridge east of the main Valdez glacier, at an elevation of 3,500 feet. The ore vein varies from 1 to several feet in width, strikes northwest and dips 70° to 80° NE. It was originally opened with a shaft and crosscut to one of the upper levels. Since that time a crosscut on the level of the mill has been driven 742 feet to tap the vein at the 300. Levels are now opened on the 50, 100, 150, and 300 foot points, all the ores passing down the main raise to the crosscut level, whence it is trammed to the mill.

At the mill the ore passes over a 1½-inch grizzly, the oversize going to a 7 by 9 inch Blake crusher, driven by a 10-horsepower Foos gasoline engine. From the bins the ore is fed to a 5-stamp Hendy mill by a Challenge feeder and is crushed to 40 mesh. The stamps weigh 1,000 pounds each, have a 6-inch drop, and fall about 110 times a minute. The pulp flows over amalgamating plates to a Diester table, which is driven by a 3-horsepower gasoline engine.

Power is furnished to the stamps by a 20-horsepower Foos engine. Another 20-horsepower Foos drives a 9 by 11 inch compressor to furnish air to the machines underground.

REYNOLDS ALASKA DEVELOPMENT CO.

The company operates the Iron Mountain property at Horseshoe Bay, on Latouche Island. The lode is a large low-grade copper-bearing ore body which strikes N. 30° E. and dips 65° W. It has been opened by three drifts and a 2-compartment vertical shaft, which is down 110 feet. A hydroelectric plant developing 100 kilowatts supplies power for the hoist, compressor, pump, and lighting.

SEA COAST MINE.

A 10-stamp mill, hydroelectric plant, and aerial tramway were erected at the Sea Coast gold property on Shoup Bay by parties who had a bond on the ground but who gave up their option late in the season.

SEALEY-DAVIS MINING CO.

The company owns a group of 13 lode claims, bordering on the eastern shore of Shoup Bay, about 14 miles from Valdez. The ore

body, a gold-bearing fissure vein which cuts a slate-graywacke series, strikes N. 50° W. and dips 61° SW. It has been opened by a 60-foot open cut, two drifts, and a crosscut, giving a total depth of about 450 feet on the vein. The average width of the vein throughout the workings is about 42 inches.

SWEEPSTAKE MINING CO.

The Sweepstake group is situated at the upper end of College Fiord, a long arm at the northern end of Port Wells. The company has run a drift 150 feet on the gold-bearing quartz lode, which varies from 18 inches to 4 feet in width. The strike is N. 65° W., the dip practically vertical, and the elevation of the drift 750 feet. The country rock is slate and graywacke.

An 1,800-foot balanced aerial tram and a mill were under construction at the property this fall. The buckets will have a capacity of 500 pounds each. The mill is a two-stamp, three-face discharge Hammond Manufacturing Co. plant, the stamps weighing 1,250 pounds each. The pulp will be passed over amalgamating plates to concentrating tables.

THOMAS-CULROSS MINING CO.

The company is operating a group of six lode claims on Culross Island, at Thomas Bay, though the work this year has been greatly hampered by litigation. The vein is quartz, carrying gold, silver, chalcopryrite, and galena, and varies in width from 1 to 5 feet. It is nearly vertical and strikes N. 15° E. A number of open cuts have been made along the outcrop and a crosscut driven 150 feet to intersect the vein along which a drift has been run 250 feet. This season the company has been installing a 10-foot Lane mill on the beach and a tram to connect the mine and plant, which will be driven by water power.

THREE MAN MINING CO.

The company owns about 40 lode claims, tributary to Landlock Bay. The main group, known locally as the Dickey claims, is at the head of the bay; the Alaska Commercial group is a little to the west of these, and the Montezuma group is on Copper Mountain.

The ore bodies at the Dickey group lie in shear zones in a slate-graywacke-greenstone series, have a general west-northwest strike, and dip 45° to 90° north. They have been opened on five levels, with over 2,000 feet of development.

The ore is carried on a short jig-back aerial tram from the lower openings to the bunkers on the wharf. The bunkers have a capacity of 800 tons..

VALDEZ GOLD CO.

The Cameron-Johnson Gold Mining Co. has been reorganized as the Valdez Gold Co. The property consists of 23 lode and 3 placer claims, about 4½ miles from Shoup Bay in an air line, and 20 miles from Valdez by way of Shoup Bay. The claims are situated in a glacial valley formerly occupied by an arm of the Shoup Glacier.

A number of veins have been discovered on the property, both in the valley and on the ridge above it, and a deep crosscut has been started to intersect these. An aerial tram and 10-stamp mill have been installed.

VALDEZ MINING CO.

The Valdez Mining Co. let contracts for the performance of the assessment work on its property, which is several miles north of Valdez, on the western side of the Valdez Glacier.

YUKON BASIN.^a

FAIRBANKS DISTRICT.

The Fairbanks district was visited during June and July and 81 placer mines and 10 quartz mines were inspected. There were 650 men employed. In addition to the mines visited there were several other smaller operations on creeks in the district which would possibly bring the number of men employed in the mines up to 800. The value of the gold output was \$1,775,000, of which \$1,725,000 was recovered from the placer mines and about \$50,000 from the quartz lodes. The distribution of the output was about as follows:

Chatanika, Cleary Creek, and tributaries.....	\$325,000	
Fairbanks Creek.....	200,000	
Little Eldorado.....	120,000	
Dome Creek.....	175,000	
Pedro and Twin Creeks.....	225,000	
Goldstream and Engineer Creeks.....	300,000	
Ester, Gold Hill, Happy, and St. Patrick Creeks.....	250,000	
Vault Creek.....	40,000	
Gilmore Creek.....	40,000	
Smallwood, Fish, Big Eldorado, and other local creeks.....	50,000	
Gold from quartz lodes, Fairbanks region.....	50,000	
		\$1,775,000
Other districts whose trade and gold are handled through the Fairbanks banks:		
Hot Springs.....	\$750,000	
Tolovana.....	700,000	
Tenderfoot, Kantishna, and outlying districts.....	75,000	
		1,525,000
Antimony, tungsten, and other metals, about.....	200,000	
		200,000
Total mineral output.....		3,500,000

During 14 years the Fairbanks district has produced gold to the value of about \$69,800,000, of which the placer mines have produced \$68,740,000. Lode mining was begun in 1910 and since that time has produced gold to the value of \$1,118,000. Other

^a Report of the Territorial mine inspector for the calendar year 1916, pp. 31-40.

minerals produced in the district are antimony, tin, scheelite or tungsten, silver and lead, having a total value of about \$312,340. Antimony was first produced commercially during 1915 and scheelite during 1916. The first shipment of lead ore was made during 1916. Tin has been recovered from the gold placers in the Hot Springs district since 1913.

During the mining season of 1916, 1,460 tons of antimony ore was shipped. The larger part of this was recovered from three mines, namely, the Eagle mine, at the head of Treasure Creek; the Chatam mine, near the head of Chatam Creek; and A. Friedriche's mine, on Vault Creek. Several other properties were operated in a small way during the first half of 1916. The fall in the price of antimony during the mid-summer put an end to most of the antimony operations.

During the year 1916, 255 tons of scheelite ore, containing over 5 per cent tungsten oxide, were shipped by freight and 6 tons of scheelite concentrate, containing about 65 per cent tungsten oxide, were shipped by parcel post. The scheelite came from three properties.

Two small shipments of lead ore were made, totaling 24 tons, containing 3.67 tons metallic lead and 1,083 ounces of silver. These shipments were made from a property situated near the head of Cleary Creek.

Several small auriferous lode mines were operated in the Fairbanks district during 1916, and gold to the value of about \$50,000 recovered. Development work was continued on a number of properties. The completion of the Government railroad and the establishing of an experimental station by the Bureau of Mines to assist in the development of the mineral industry will stimulate an interest in lode mining that is likely to develop several good producing lode mines.

TOLOVANA DISTRICT.

The Tolovana district was visited during July. Seventeen properties were being operated at the time of visit. The gold-bearing gravels are largely slate, chert, and quartz. All of the 17 properties visited were on the benches of Livengood Creek, which extended from opposite No. 1 below Discovery to No. 17 above Discovery, a distance of about 3 miles. There were several smaller operations on Lillian, Ruby, Amy, and Olive Creeks and prospecting on Mike Hess Creek. All of the smaller operations were closed at time of visit on account of need of water for sluicing caused by lack of rain. The gold from the deep channels is dark colored and has an assay value of \$18.75 to \$18.90 per ounce. The total value of the gold production for this district during the year was about \$700,000. Gas is encountered in the deep placer mines of this district. One man lost his life on Mike Hess Creek and three were burned by the ignition of gas on Livengood Creek.

Timber is plentiful along the Tolovana River. Two sawmills situated at West Fork furnish all of the lumber needed in this district. The timber on Livengood Creek is small and scrubby. About 250 men were employed in this district.

HOT SPRINGS DISTRICT.

The Hot Springs district was visited during July. About 16 placer mines, employing some 300 men, were operated in this district during the year. These yielded gold to the value of about \$750,000. Of this Eureka Creek produced about \$60,000, Wood-chopper, Sullivan, and American Creeks, and their tributaries produced \$690,000. In addition to gold 70 tons of tin or cassiterite ore were recovered and 58 tons shipped as a by-product from the placer mines operating for gold. The value of the tin ore was approximately \$35,000, making the total mineral production of the district \$785,000. The gold production for 1915 was \$610,000.

CIRCLE DISTRICT.

The Circle district includes the Birch Creek district and the placers of Woodchopper and Beaver Creeks. It is estimated that gold to the value of \$250,000 was produced from about 50 mines, employing some 200 men. The largest output came from Mammoth Creek, where a dredge was operated during the entire mining season. Hydraulic mining operations were continued on a number of creeks.

RAMPART DISTRICT.

Mining continued in the Rampart district on about the same scale as in recent years on Hunter, Little Minook, Hoosier, Slate, Big Minook, and Ruby Creeks. The district produced approximately \$30,000.

EAGLE DISTRICT.

Mining operations were continued in the Eagle and Seventy-mile districts on about the same scale as in the previous years. About \$20,000 was produced during the season 1916.

RUBY DISTRICT.

The Ruby district was visited in August and 16 properties inspected. The principal creeks are Poorman, Flat, and Spruce, in the Poorman district; and Long Creek, in the Long Creek district. Early in the season a large increase in the gold output was anticipated in this district, but the pinching, or decrease of the gold values in the pay gravel, in some of the most promising claims on Long Creek, below the margin of profitable mining, together with the extended drought during the first part of the season, curtailed the output considerably. The output for 1915 was \$800,000. The estimated output for 1916 was \$840,000, an increase of \$40,000 over the production for 1915. The completion of the new dredge on Greenstone Creek early in the season helped materially to maintain the output. New deep-underground placer mining operations on Birch and Straight Creeks (one plant on each creek) showed considerable cassiterite or tin ore to be associated with the placer gold, but no effort was made to save it. It is not known, therefore, if there is enough to be mined commercially. The bedrock formation is granite; the overburden is 80 feet in depth, frozen. Some cassiterite ore was recovered from the placer-gold operations on Midnight Creek and shipped to Seattle. On Trail, Tamarack, Bear Pup, Tenderfoot, and Duncan Creeks mining operations were continued during the open season as in former seasons. About 300 men were employed.

The Alaska Road Commission expended during the open season about \$70,000 on the Ruby-Long wagon road, which will be of inestimable value to the operators and to the development of this district. The closest mines are about 30 miles from Ruby, the nearest landing point on the Yukon River where supplies are landed. The roads in the summertime have been almost impassable, resulting in freight charges being as high as 8 and 10 cents per pound from Ruby to Long City.

MARSHALL DISTRICT.

The Marshall district is on the Yukon near its delta, and about 50 miles below Russian Mission. Gold was found in this district in 1913, but until 1916 the production was very small. In 1916 some rich placers were developed on Willow Creek, on which seven plants, employing about 200 men, were operated and produced gold to the value of \$250,000. There was a great influx of prospectors to this district before the close of the summer season. The district should get a thorough prospecting during the winter of 1916-17, and some very good discoveries of placer gold may be expected. A hydraulic plant was being installed on Elephant Creek, and some work was done on Disappointment Creek, and a small amount of gold recovered. Freight and supplies for the mines are shipped via the Yukon River to Marshall.

Marshall is the principal distributing point, and is situated about 12 miles from Willow Creek. From Marshall supplies are transhipped by small boat or launch through the sloughs of the river delta to within 3 miles of the Willow Creek camps, where they are loaded on wagons and hauled along one of the poorest wagon roads in the Territory to the mines on Willow Creek. Some development work was done on the quartz claims near the head of Willow Creek.

KOYUKUK DISTRICT.

The Koyukuk district was not visited during 1916 by the inspector. The estimated production of gold for this district this season is \$300,000. Of this amount, about \$200,000 was taken from the deep placers of Hammond and Nolan Creeks. Linda, Gold Myrtle, and other smaller operations contributed their quota to the output. Two drills were shipped to the district during the open season. The estimated production includes the Indian River and the Chandalar districts. About 200 men were employed in the district.

IDITAROD DISTRICT.

The Iditarod district was visited during the month of August. The gold production of the Iditarod district for 1916 was about \$2,000,000, an increase of \$150,000, over the production in 1915.

Eleven placer mines, four steam-scraper plants, and three dredges were operated. One dredge was on Flat Creek, and one was on Otter Creek. The other dredge was installed on Black Creek by the Otter Creek Dredging Co., but operated only a part of the season. It is a 2½-cubic-foot revolving-screen flume dredge, built by the Union Construction Co. Power is furnished by two 50-hp. semi-Diesel fuel-oil internal-combustion engines, built by the Scandia Engineering Co., of Stockholm, Sweden. The two 50-hp. Union distillate engines on the Otter Creek dredge were taken off and replaced with two 50-hp. hot-bulb fuel-oil Atlas engines, built in Stockholm, Sweden.

Some of the richest placer mines in the district are situated near the top of the divide and at the extreme head of Flat, Happy, and Chicken Creeks, and can be operated only during the spring thaw or very wet weather, they being so high the drainage area is very limited. During the winter of 1915-16 snow fences were constructed on top of the divide between those creeks, causing the snow to pile up in large drifts to a depth of over 80 feet, which furnished water for sluicing on those claims until the 1st of August, thereby adding materially to the gold output. The largest operations were those of the two dredges on Flat and Otter Creeks. There were approximately 400 men employed in the district.

INNOKO DISTRICT.

It is estimated that 30 mines were operated in the Innoko district in 1916, that about 150 men were employed, and that gold was produced to the value of approximately \$160,000. The principal producing creeks were Ophir, Gaines, Spruce, and Yankee. New pay was discovered on the benches, left limit of Gaines Creek, just above the mouth of Little Creek, which may prove to be of considerable importance. There was also some prospecting with drills on Moore and Yankee Creeks, for the purpose of installing dredges. A discovery of some importance was made on Boob Creek, a tributary of Madison Creek, which is a tributary of Tolstoi Creek, a tributary of the Dishna River, which flows into the Innoko River a short distance above Dishkakaket. Only a few thousand dollars were produced during 1916, but indications are that a very good output can be expected during 1917. A town named Cooper was situated at the confluence of Madison and Tolstoi Creeks, which is reached in the summer time by boats up the Innoko River to the Dishna River, and thence up the

Dishna River to opposite the town of Cooper, where a landing is made and goods hauled 5 miles across country to the town on Tolstoi Creek. Tolstoi Creek is too shallow most of the open season for navigation even by a small gas boat. There should be some provision made for the construction of wagon roads into new camps.

KUSKOKWIM BASIN.

A dredge was shipped for installation on Candle Creek, in the Takotna district, or upper Kuskokwim. It was to be shipped to McGraths during the summer season and hauled from there over to Candle Creek during the spring of 1917, but owing to the breaking of the propeller shaft of the boat that was bringing the dredge machinery from Seattle to the Kuskokwim and the boat putting back to Seattle the machinery did not arrive before the close of navigation, thus delaying the installation of the dredge for one year. Good returns are said to have been obtained from Canyon and Windy Creeks; also the Aniak district, in the lower Kuskokwim Basin; and from Candle Creek, on the upper Kuskokwim Basin. The exact amount of the gold output for 1916 has not been learned, but it is estimated to be about the same as last year, \$100,000. About 80 men were employed in the operation of mines in the Kuskokwim Basin.

SEWARD PENINSULA.

The Seward Peninsula mines produced gold to the estimated value of \$2,900,000 in 1916, which is the same as that of 1915. In addition to the gold production there were 162 tons of tin ore valued at \$81,000, 15 tons of scheelite or tungsten ore valued at \$22,000, and 70 tons of antimony ore valued at \$6,000, making a total mineral production for the Peninsula of \$3,009,000. About 70 tons of graphite was mined, but on account of lack of transportation facilities to Nome no shipments were made.

Since 1897 the Seward Peninsula mines have produced gold to the value of \$74,351,000. Most of this gold was taken from the placer mines, although from 1903 to 1907 the big Hurrah Quartz mine produced some gold and small amounts have been recovered from several lodes at different times.

The other mineral resources of the Peninsula that have been developed are the York tin deposits, first developed in 1900; the antimony deposits, first mined on a commercial scale in 1915; scheelite or tungsten ore, mined near Nome during 1916 for the first time commercially; and a small amount of copper has been shipped from Kougarok district and Pilgrim River. Some graphite has been mined in the Port Clarence and Fairhaven districts. Coal has been mined in the Fairhaven district in a small way since 1902 for local consumption. There was considerable prospecting for copper, tungsten, and gold lodes during the past year.

A corps of engineers and geologists investigated the practicability of putting in a hydroelectric power plant in the Saw Tooth Range in the Kougarok district, the purpose being to furnish power to the dredges and lode mines of the Peninsula. Investigations of the lode and gravel deposits were also made by the engineers and geologists to determine if they could dispose of sufficient power to warrant the installation of power plant. The investigations had a tendency to stimulate interest in the prospecting and development of lode mines.

There were 31 dredges operated on Seward Peninsula during the year 1916. There were dredging for gold 29 dredges and for tin two. The two tin dredges were in the York district. The gold dredges were situated as follows: Seven in the Nome district, 4 in the Solomon River, 10 in the Council district, 3 in Port Clarence district, 3 in Fairhaven, and 2 in the Kougarok district. One new dredge was installed on Glacier Creek. One of the Flodin dredges moved from Solomon River to Canyon Creek in the Casadepaga River section. Another one of the Flodin dredges was purchased by O. W. Flowers and operated for a part of the season on upper Solomon River and later in the season was dismantled preparatory to moving during the winter 4 miles lower

down Solomon River. The Mystery Creek dredge was purchased by the Northern Light Mining Co. and moved from Mystery Creek to Ophir Creek in the Council district. The Warm Creek dredge, which was idle during 1915, was changed from a revolving screen dredge to a flume dredge and operated a part of the season on Warm Creek. The dredges of the Nome Consolidated Mining & Development Co. were idle during 1916. This company has three dredges in the Nome district. The one on the third beach, or No. 8 Cooper, was dismantled during the summer and fall. Extensive alterations are planned in the reconstruction of the dredge; also the completion of the Flat Creek dredge which has been partly built ever since 1911. There were 16 idle dredges in Seward Peninsula during 1916. Some are idle for the reason that the productive placer ground has been worked out and the cost of moving a machine as large as a dredge any distance without dismantling it is prohibitive. The work of dismantling, rebuilding, and hauling over almost impassable roads is more expensive than purchasing a new dredge that is fitted to the ground, as nearly every creek or piece of dredgeable ground has different characteristics either as to depth or formation. A dredge that is suited to one piece of ground would be practically worthless on another of different formation and character. One of the most important things in building a dredge is to know the depth and character of the ground desired to be dredged. It may mean the difference between success and failure of a gold-dredging enterprise.

About 65 deep placer mines and 30 open-cut plants were operated on the Peninsula during 1916. The placers developed on Dime Creek, a tributary of the Koyuk River in the Council district, which was discovered during 1915, have yielded gold to the value of \$150,000 this year. Most of this was extracted during the winter. Reported rich strikes on the benches of this creek during the summer have caused added activity to this camp and a material increase in the gold output may be expected for 1917. Sweepstake Creek, another tributary of the Koyuk River, has been a steady producer for several years and still adds its quota to the gold output of the Council district. The 10 dredges in the Council district have produced about \$600,000. Placer mining in the Kobuck region during 1916 continued. The output was about \$15,000.

The mines of Seward Peninsula employ about 1,100 men. It is very hard to get the exact number for the reason that there are so many engaged on small open-cut or hydraulic operations in remote parts of the Peninsula, where it is impracticable for the inspector to visit. In view of the vast territory to be covered in a limited season, he is obliged to confine his inspection to the principal mining centers. These small operations are, however, conducted on such a scale as to involve little risk of serious accidents.

There was an abundance of rainfall during the mining season which was favorable to hydraulic mining and helped to increase the output of the larger hydraulic operations as well as to stimulate operations on a number of smaller creeks which are neglected during dry seasons.

The inaccessibility of a large part of the Peninsula through lack of transportation facilities is very discouraging to prospectors. While the Alaska road commission has done good work with the small amount of funds allotted to it, yet there should be more and better wagon roads. This is of the greatest importance to the development of the mineral resources of the Peninsula. A wagon road from Nome to Candle through the center of the Peninsula would be of inestimable value. The cabins erected along the trail between Dahl Creek, in the Kougarok, and Candle with the moneys received from the Territorial fund were a great benefit to the traveling miners coming to Nome to take the last boats to the States.

On account of the extremely stormy weather during the latter part of 1916 no boats were able to sail to the Kotzebue Sound, as only small gasoline launches make the trip with the exception of one or two freight steamers which take in supplies during the summer and they are not allowed to take passengers, although some of them have

accommodations for 50 or more. The custom officers, however, will not allow steamships to take passengers when they have gasoline aboard and that was a part of the steamers' cargo. This was being supplied to the dredges in that locality. Peculiarly, though, a small gasoline launch, loaded with gasoline from stern to stern and hardly seaworthy, can take all the passengers that can climb aboard. As many as 52 have been on launches that did not have accommodations for 10. But such launches are welcome conveyances compared to walking across the country, it taking three days to walk from one road house to another. Some provision should be made allowing large steamships on that northern run to take passengers between Nome and those extreme northern towns.

A road from Davidson's landing, on Marys River, to Taylor Creek would be of great value to the upper Kougarak. The road is already there but is impassable during the summer season. A very good road has been constructed by the Alaska Road Commission along the right limit of Solomon River from the ocean beach to East Fork, on Solomon River, a distance of about 12 miles. With very little added expense and work a good wagon road could be built from East Fork to Council and along the beach from Bonanza River to Nome, which would serve as an auto-truck road and greatly benefit the mining industry of that section. The road around Cape Nome was built during the summer of 1915 from the money received from the Territorial forest fund and has been of great assistance to the operators of the Solomon River and Council districts during the summer, as the extremely stormy weather made travel by boat from Nome to Solomon impossible most of the season. Owing to dredges being made up of very heavy machinery, the moving of them from one locality to another is quite impossible without roads. The construction of a wagon road, therefore, from Candle to the Kugruk River is very much needed, as the coal fields located there are of importance to the operators.

ACCIDENTS.

A fairly accurate estimate of all the men working about mills, dredges, quarries, lode, and placer mines in the Territory gives a total of 9,118, of whom 4,194 are employed in the lode mines, mills, and quarries and 4,924 about the placer mines and dredges. In the lode mines approximately 2,796 are underground and 1,398 are surface employees. In the placers the number of both surface and underground men approximate 50 per cent of the whole. Of the surface employees at the lode mines, about 925 were engaged in mills or metallurgical plants.

The mining operations throughout the Territory are so scattered and the facilities for traveling so limited that it is impossible for the Federal inspector's office, with its meager personnel, to cover the districts more than superficially, hence the collection of data relating to mine accidents depends largely on the care of the operators in sending in reports. In presenting data thus gathered, there is a tendency to exaggerate the number of accidents at mines under the control of operators who report all accidents and to minimize those at mines where the operator neglects to make the proper reports.

The information covering fatal accidents is fairly accurate for all branches of the industry; that for serious accidents is representative for the lode mines but poor for the placers; and that for slight

accidents fair for the lode properties but absolutely lacking for the placers. For the sake of greater accuracy in tabulating this information, the totals for serious and slight accidents are taken from the annual reports, which differ slightly from the totals of the individual reports. The accidents were distributed as follows:

Accidents at mines in Alaska during 1916.

Fatalities:

Lode mines—	
Underground.....	19
Surface.....	3
Total lode.....	22
Metallurgical plants.....	0
Placer mines—	
Underground.....	5
Surface.....	2
Total placer.....	7
Total fatalities.....	29

Serious accidents:

Lode mines—	
Underground.....	116
Surface.....	450
Total lode.....	166
Metallurgical plants.....	18
Placer mines—	
Underground.....	15
Hydraulicking.....	2
Dredging.....	7
Total placer.....	24
Total serious accidents.....	208

Slight accidents:

Lode mines—	
Underground.....	331
Surface.....	6135
Open pit.....	1
Total lode.....	467
Metallurgical plants.....	109
Placer mines.....	
Total slight accidents.....	576

The accident ratios, figuring the lode mines as operating the entire year, and the placer mines as operating 6 months, are as follows:

In lode and placer mines.....fatalities per 1,000 men employed..	3.180
In lode mines, underground.....do.....	6.795
In lode mines, surface.....do.....	2.146
In placer mines, underground.....do.....	4.062
In placer mines, surface.....do.....	1.625

^a Exclusive of metallurgical plants.

In lode mines, underground.....	serious per 1,000 men employed..	41.487
In lode mines, surface (exclusive of metallurgical plants).....	do.....	105.708
In metallurgical plants.....	do.....	19.459
In lode mines, underground.....	slight per 1,000 men employed..	118.383
In lode mines, surface (exclusive of metallurgical plants).....	do.....	285.412
In metallurgical plants.....	do.....	117.838

The ratios for serious and slight accidents at the placer mines are not given, as they would be misleading on account of their incompleteness.

It is impossible to place the exact responsibility for mine accidents. The classification used by the inspectors of the Union of South Africa has been adopted for this report, as shown in the following table, the percentage error of which is comparatively small:

Responsibility for accidents in Alaska during 1916.

Item No.	Cause of accident.	Fatal.	Serious.
		<i>Per cent.</i>	<i>Per cent.</i>
1	Danger inherent to work or misadventure.....	51.724	42.038
2	Defective plant or material.....	3.448	6.309
3	Fault of injured person:		
	(a) Carelessness.....	20.69	40.127
	(b) Ignorance.....	3.448	
	(c) Disobedience of orders.....	6.897	.637
4	Fault of management.....	10.345	1.274
5	Fault of foremen.....		1.274
6	Fault of others (fellow workmen).....	3.448	5.096
7	Joint fault of items 3, 4, 5, and 6.....		3.185
	Total.....	100	100

The proportion of accidents due to carelessness, both fatal and serious, is shown to be very large. Large as these figures seem, they are probably too small, as many of the accidents due to falls of rock had to be classified under "Danger inherent to work," whereas if complete data had been available it would be shown that they could have been avoided by proper caution. If foremen and other men occupying responsible positions were required to hold certificates of competency which could be revoked for cause, the accidents due to careless methods would be minimized.

RECOMMENDATIONS.

The following recommendations are offered as a means of reducing the number of accidents and improving conditions in the mineral industry throughout the Territory:

RULES GOVERNING 10-ACRE LEASES NEEDED.

Rules and regulations governing the operation of mines on land covered by the free-use 10-acre permits should be issued and before a permit is granted the applicant should supply evidence that he is familiar with the operation of coal mines and their ventilation or that he will place a competent man in charge.

APPROPRIATION FOR SURVEYING COAL LAND NECESSARY.

Isolated tracts of coal lands not covered by present surveys should be surveyed. Under the present law an operator can obtain only 10 acres of coal land outside the surveyed areas and this under the restriction that the coal mined shall be used locally. A specific instance of the need of surveys is afforded by the Happy Valley Coal Co., which obtained a free-use 10-acre permit to mine coal on a tract of land near Bluff Point, on Cook Inlet, and made application for a larger lease that it might open a mine large enough to supply coal to empty freight vessels returning to Portland, Oreg. The application had to be denied as there were no funds available for a survey, although these surveys are authorized by the coal leasing act of 1914. In consequence the company was restricted to a local market, vessels were deprived of return freight, and the coast market denied a cheap fuel.

BUREAU OF MINES INSTRUCTION SHOULD BE EXTENDED TO ALASKA.

The Bureau of Mines should have specific authority to assign first-aid miners to the Territory of Alaska in order to offer the miners of this Territory the same advantages as those in the States. The remoteness of many of the mines and the lack of facilities for rapid travel make this doubly necessary.

DEPUTY INSPECTORS SHOULD BE APPOINTED.

The appropriation to the Bureau of Mines for mine inspection in Alaska should be increased to provide for office rental, the purchase of technical equipment, and the employment of at least four deputy inspectors.

TABLE 1.—*Accidents in all Alaskan metal mines and metallurgical plants during the year ended Dec. 31, 1916.*

METALLURGICAL PLANTS (ORE-DRESSING AND MILLING).			
	Killed.	Seriously injured. (Time lost, more than 14 days.)	Slightly injured. (Time lost, 1 to 14 days.)
Number killed or injured by —			
1. Haulage system (cars, motors, etc.).....		3	14
2. Railway cars or locomotives.....			1
3. Crushers, rolls or stamps.....		5	19
4. Tables, jigs, etc.....			1
5. Other machinery.....		3	15
6. Falls of persons.....		2	4
7. Suffocation in ore bins.....			
8. Falling objects (rocks, timbers, etc.).....			6
9. Cyanide or other poisoning.....			1
10. Scalding (steam or water).....			1
11. Electricity.....		1	
12. Hand tools, axes, bars, etc.....		2	6
13. Nails, splinters, etc.....			6
14. Flying pieces of rock from sledging or crushing.....			2
15. Other causes.....		2	23
Total number killed or injured at mills.....		18	109

TABLE 1.—*Accidents in all Alaskan metal mines and metallurgical plants during the year ended Dec. 31, 1916—Continued.*

METAL MINES.

	Killed.	Seriously injured. (Time lost, more than 14 days.)	Slightly injured. (Time lost, 1 to 14 days.)
<i>Underground.</i>			
Number killed or injured by—			
1. Fall of rock or ore from roof or wall.....	10	25	59
2. Rock or ore while loading at working face or chute.....		19	47
3. Timber or hand tools.....		14	22
4. Explosives.....	3	8	5
5. Haulage system (mine cars, mine locomotives, breakage of rope, etc.).....		17	40
6. Falling down chute, winze, raise, or stope.....	3	8	14
7. Run of ore from chute or pocket.....	1	4	13
8. Drilling accidents (by machine or hand drills).....		15	30
9. Electricity.....			3
10. Machinery (other than locomotives or drills).....		1	5
11. Mine fires.....			
12. Suffocation from natural gases.....	1		3
13. Inrush of water.....			
14. Nails, splinters, etc.....		1	10
15. Other causes.....		16	76
Total number killed or injured underground.....	18	128	327
<i>Shaft accidents.</i>			
Number killed or injured by—			
16. Falling down shafts.....	3		
17. Objects falling down shafts.....			1
18. Breaking of cables.....			
19. Overwinding.....			
20. Skip, cage, or bucket.....	2	1	2
21. Other causes.....	1	2	1
Total number killed or injured by shaft accidents.....	6	3	4
<i>Surface accidents. (At surface yards and shops.)</i>			
Number killed or injured by—			
22. Mine cars or mine locomotives, gravity or aerial trams.....			7
23. Railway cars and locomotives.....	1	10	8
24. Run or fall of ore in or from ore bins.....			2
25. Falls of persons.....	1	9	8
26. Nails, splinters, etc.....		1	9
27. Hand tools, axes, bars, etc.....	1	1	22
28. Electricity.....	1	1	3
29. Machinery.....		5	11
30. Other causes.....	2	23	65
Total number killed or injured by surface accidents.....	5	50	135
<i>Open-pit accidents.</i>			
Number killed or injured in pit by—			
31. Falls or slides of rock or ore.....		1	1
32. Explosives.....			
33. Haulage system (cars, locomotives, etc.).....			
34. Steam shovels.....			
35. Falls of persons.....			
36. Falls of derrick, booms, etc.....			
37. Run or fall of ore in or from ore bins.....			
38. Machinery (other than locomotives or steam shovels).....		3	
39. Electricity.....			
40. Hand tools.....		1	
41. Other causes.....		4	
Total number killed or injured by open-pit accidents.....		9	1
Grand total, metal mines.....	29	190	467

NOTE.—The number of widows reported as due to the 29 fatalities was 6, whereas the number of orphans under 16 years was 24. Of the 208 serious injuries, 3 were permanent total disability and 13 permanent partial disability. The slight injuries at the placer mines were not reported, hence are not included in this table.

TABLE 2.—*Fatalities in lode mines in Alaska*

Date of accident.	Name of person.	Nationality.	Occupation.	Age.	Married or single.	Dependents.	
						Widow.	Children under 16 years.
Jan. 3	Eugene M. Kelly.....	American	Machineman..	40	Married...	Yes.....	3
4	Domenico Ciotta.....	Italian	Miner.....	41	do.....	do.....	4
4	Guio Guizzo.....	do.....	do.....	29	Single.....		
16	Ernest Gost.....	Greek.....	do.....	26	do.....		
Mar. 17	James Gunn.....	English.....	Driver.....	33	do.....		
21	Emil Oswald Heinze.....	German.....	Mucker.....	50			
22	Joseph Vizzetti, sr.....	Italian.....	Machineman..	51	Married...	Yes.....	5
31	Gus P. Clarey.....	American.....	do.....	26			
Apr. 19	Arnold C. Bonsdorff.....	Dane.....	Laborer.....	34	Single.....		
27	Joe Kosich.....	Austrian.....	Machineman..	26	do.....		
July 1	Oscar Johnson.....	Swede.....	Shift boss.....	33	do.....		
13	Nik Krinis.....	Greek.....	Bulldozer.....	47	Married...	Yes.....	5
15	Rune Carlson.....	American.....	Motorman.....	22	Single.....		
Aug. 4	Nick Olson.....	Russian.....	Skip tender.....				
7	Christofer Christoferson.....	Norwegian.....	Miner.....	26	Single.....		
13	Fred Mattson.....	Finlander.....	Tramper and cager.	23	do.....		
15	Joseph Gretland.....	Norwegian.....	Miner.....	28			
Sept. 6	Ole Roen.....	do.....	Machineman..	22	Single.....		
Oct. 25	Gus Vos.....	Hollander.....	Steel worker..	30	do.....		
26	A. Sheboff.....	Russian.....	Mucker.....	23	do.....		
Nov. 30	Louis Tremantine.....	Italian.....	Miner.....	49	Married...	Yes.....	5
Dec. 14	H. D. McClellan.....	American.....	Lineman.....	37	do.....	do.....	2

TABLE 3.—*Fatalities in placer mines in Alaska*

Date of accident.	Name of person.	Nationality.	Occupation.	Age.	Dependents.
.....	Alex Woodside.....	Scotchman.....	Operator.....		None.....
May 10	Michael Pasich.....	Hungarian.....	Mucker.....	24	do.....
July 2	Chris Heeney.....	Irish.....	Operator.....	42	do.....
11	Albert Bjorklund.....	Scandinavian.....	do.....	30	do.....
Aug. —	Jack Seaberg.....	do.....	do.....	40	do.....
—	Oscar Erickson.....	Swede.....	Prospector.....		
Oct. 11	Andros Peter Theodore Anderson.....	Norwegian.....	Engineer.....		None.....

during the year ended Dec. 31, 1916.

Name of company and mine.	Nature and cause of accident.
Alaska Juneau Gold Mining Co.; Alaska Juneau. Alaska Treadwell Gold Mining Co.; Treadwell.	Back broken by dirt slide while timbering portal. Clotta and Guizzo had been bulldozing in stope and started to return to gangway with powder. It had been their custom to leave bunches of powder tied with primers at head of manway raise, and in passing this powder it exploded, throwing both men into raise and killing them.
Kennecott Copper Corporation; Bonanza. Alaska United Gold Mining Co.; 700-Foot. Kennecott Copper Corporation; Bonanza. Alaska Gastineau Mining Co.; Perseverance. Alaska United Gold Mining Co.; 700-Foot. Alaska Juneau Gold Mining Co.	Walked over chute being drawn and was carried down with ore and smothered. Killed by fall of loose rock in drift while showing a friend through mine. Crushed by descending timber skip while walking up incline at noonhour. Crushed by slab falling from roof while operating machine drill. Killed by fall of rock from pillar and roof while barring down.
.....do.....	While ascending stairway next tramway for hauling construction material, injured placed hand on passing car and caught either hand or sleeve, which threw him under car and crushed him.
Ellamar Mining Co.; Ellamar	Standing above chute being drawn and was pulled down with ore and buried. Injured in repairing pipe line in manway; allowed end of timber he was using for lever to project into hoisting compartment. Descending cage struck this timber and crushed his head against shaft timbers.
Alaska United Gold Mining Co.; Ready Buillon. Alaska Gastineau Mining Co.; Perseverance. Chichagoff Mining Co.; Chichagoff..... Alaska Industrial Co.; Jumbo..... Ellamar Mining Co.; Ellamar	Had blasted hole in stope to break down slab. On returning to stope a second slab fell and crushed him. Fell into oreway; probably was attempting to bar large rock out of car. Thrown out of skip, which was off track, and fell down shaft. Fell into oreway. Fell into shaft.
Mount Andrew Mining Co.; Mount Andrew. Alaska Juneau Gold Mining Co.do.....	Gretland went into another man's place after the man had gone to notify foreman it was unsafe. A rock fell while Gretland was trying to place a timber and killed him. Asphyxiated by powder gas in raise. Crushed by steel plates sliding from car which he was unloading on inclined tramway.
Alaska Gastineau Mining Co.; Perseverance. Kennecott Copper Corporation; Beatson. Alaska Gastineau Mining Co.; Perseverance.	Crushed by fall of rock from side of drift while cleaning up around chute. Crushed by fall of slab from roof of stope while running machine drill. Buried beneath snow slide.

during the year ended Dec. 31, 1916.

Name of operator and mine.	Nature and cause of accident.
Watson & Woodside.....	Crushed in the shaft bottom by falling bucket, due to cable breaking.
Gold Hill Mining Co.	While driving drift in frozen gravel, unexpected thawed ground ran and buried him.
Heeney, Pike & Miller..... Bjorklund & Bachner.....	Crushed by fall of gravel from roof of drift. Deceased while adjusting steam points in bottom of shaft was weakened from gas from decayed vegetation. While being hoisted he fell from chair.
Seaberg & Isler..... Oscar Erickson.....	Killed by falling boom. Overcome in bottom of shaft by fumes from fire set to thaw face of drift.
C. E. Kimball Co.	Fell from scow crossing from river bank to dredge; drowned.

TABLE 4.—*Serious accidents on surface about lode mines*

Date of accident.	Name of person.	Nationality.	Occupation.	Age.
Jan. 6	Earl Denham	American	Drill sharpener	30
20	Chris Johnson	Norwegian	Laborer	24
Feb. 7	Ben Dahl	do	Feeder	25
10	Joseph F. Zace	American	Laborer	43
12	Mike Moor	Belgian	do	42
15	George Petersen	Swede	do	32
Mar. 4	John Mackanen	Finlander	Millman	46
15	George Martin	Hungarian	Chuteman	30
18	George Atkinson	English	Laborer	32
20	Chester Leonhardt	American	Millman	19
20	Fred Christensen	Norwegian	Laborer	25
28	J. E. Steers	American	Fireman	46
Apr. 2	Thomas Murphy	do	Laborer	30
2	Peter Viano	Italian	do	28
20	F. J. Samson	German	Mucker	51
May 3	Ed R. Evans	American	Crusher	27
6	B. M. Rice	do	Laborer	21
7	Charles Slaymaker	do	do	40
10	G. J. Cassidy	do	Donkey engineer	46
17	John McCarthy	do	Mucker	32
24	E. J. Sliter	do	Laborer	46
29	Bob Rosandich	Austrian	do	25
31	Einas Solund	Finlander	Mucker	21
June 5	David A. Wheeler	American	Laborer	29
12	Lou McDonald	do	do	31
15	Herman Hugh	Swiss	do	22
Aug. 3	Otto Stewart	do	Steel sharpener	38
18	A. A. Squires	do	Flume tender	65
Sept. 4	Tury Saks	Russian	Laborer	36
7	Jess Coddington	American	Gripman	23
20	George N. Peterson	Dane	Mill oiler	24
Oct. 2	E. I. Whitmore	American	Skip operator	40
23	Olaf Jamne	Norwegian	Painter	40
30	Battista Bobzadelli	Italian	Concentrator	30
Nov. 19	Frank Ross	American	Electrician	25
Dec. 19	Steve Vukovich	Montenegrin	Mucker	27

in Alaska during the year ended Dec. 31, 1916.

Name of company and mine.	Nature and cause of accident.
Alaska Mexican Gold Mining Co.; Mexican.	Third and fourth fingers of left hand crushed. Rested hand underneath hammer of drill sharpener and accidentally turned on the air.
Alaska Juneau Gold Mining Co.; Alaska Juneau.	Bruised and cut about head. Fell from dock while unloading lumber.
Alaska Treadwell Gold Mining Co.; Treadwell.	While changing cam shaft the shaft fell off horses and a cam slipped off, crushing big toe on right foot.
Alaska Juneau Gold Mining Co.; Alaska Juneau.	Abscess on right hand, due to infected cut.
Alaska Gastineau Mining Co.; Perseverance.	Fell while carrying steel on mill grade and broke small bone in ankle.
Alaska United Gold Mining Co.; 700-Foot.	End of index finger on left hand crushed while moving drill sharpener off track. Accidentally placed finger under hammer as machine was turned over and it fell on finger.
Alaska Gastineau Mining Co.; Perseverance.	Left ankle bruised by falling stamp while making repairs in mill.
Alaska Juneau Gold Mining Co.; Alaska Juneau.	Injured was on running board of locomotive, which backed up against train, catching left foot between drawheads and bruising it badly.
Alaska Gastineau Mining Co.; Perseverance.	Resting hand on bowlder when one rolled down from above and broke two bones in right hand.
Alaska Juneau Gold Mining Co.; Alaska Juneau.	Attempted to stop moving belt; left arm was jerked between pulley and belt and broken.
Alaska Treadwell Gold Mining Co.; Treadwell.	Struck by broken cable on donkey engine; sustained a broken collar bone and two scalp wounds.
Alaska United Gold Mining Co.; Ready Bullion.	Burned on right eye by hot scale while cleaning boiler.
Alaska Juneau Gold Mining Co.; Alaska Juneau.	Riding on load of baled hay when one bale struck platform and threw him onto rocks below. Right arm fractured, right hip sprained, and head cut.
Alaska Gastineau Mining Co.	Fell from platform while throwing down timbers and fractured skull.
Alaska Treadwell Gold Mining Co.; Treadwell.	Bruises on back, internal injuries. Preparing blast on mill grade when a rock rolled down from above and struck him on back.
Alaska Juneau Gold Mining Co.; Alaska Juneau.	Middle finger of left hand cut and bruised at end. Injured was prying piece of steel from grizzly plate when bar slipped, causing him to fall and catch his finger between the bar and a bolt-head.
Alaska Gastineau Mining Co.	Left leg broken at ankle. Injured attempted to shift belt from high-speed to intermediate pulley with his foot and was caught between belt and pulley.
Alaska Treadwell Gold Mining Co.; Treadwell.	One bone in left leg broken. Caught between drawbar of locomotive and car while attempting to couple them.
Alaska Juneau Gold Mining Co.; Alaska Juneau.	Left ankle caught between planks and bruised while moving donkey engine from wharf onto barge.
Alaska Gastineau Mining Co.	Right ankle bruised. While pushing car the car bumped at end of track and backed a foot or two; injured fell on track and car-bumper passed over ankle.
Alaska Gastineau Mining Co.	Ankle and foot bruised by falling plank while working pile driver.
Alaska Gastineau Mining Co.	Right leg bruised. Injured was trying to put dump car on track when car slipped and fell on him.
Alaska Gastineau Mining Co.	Both ankles bruised. Injured was thrown over dump by car turning over as he attempted to dump it.
Alaska Gastineau Mining Co.	Left arm crushed; amputation necessary. While helping change cable from bottom to top of drum on small hoist at incline tramway, arm was caught in loop of cable.
Alaska Gastineau Mining Co.	Right thumb bruised while loading steel on truck.
Alaska Free Gold Mining Co.	Mitten caught in saw. Palm cut.
Alaska Gastineau Mining Co.; Perseverance.	Struck left hand with hammer, causing slight abrasion which became infected.
Alaska Gastineau Mining Co.	Cut over left eye and body bruised. Thrown from car when horse shied, throwing car off track.
Alaska United Gold Mining Co.	Two bones broken in left leg and chest bruised. Shifting belt from line-shaft pulley; caught in belt and thrown to floor.
Kennecott Copper Corporation; Jumbo.	Fracture, left arm. Injured stopped tramway to tighten bucket grip; fastened bucket with rope which failed to hold when grip was loosened, and was knocked to the ground.
Kennecott Copper Corporation	Lost third, fourth, and fifth fingers and portion of left hand. Fingers drawn into bevel gear when injured attempted to put grease on gear with his hand.
Alaska Gastineau Mining Co.	Right elbow dislocated, right hand bruised, and left hand so badly crushed that amputation was necessary. Injured was moving conveyor-dumper when his hands caught between rope and "niggerhead" on "gypsy."
Alaska Juneau Gold Mining Co.	Left arm broken at shoulder and wrist by fall from roof of mill.
Alaska Gastineau Mining Co.	Head, jaw, and wrist bruised while attempting to move belt from pulley.
Alaska Juneau Gold Mining Co.	Sprained right ankle. Slipped on mat at foot of stairs in mill.
Alaska Juneau Gold Mining Co.	End of thumb on right hand crushed. Injured attempted to remove rock from wheel which he used to block the car and the wheel pinched his thumb.

TABLE 5.—*Serious accidents underground in lode*

Date of accident.	Name of person.	Nationality.	Occupation.	Age.
Jan.	4 Joseph Succa.....	Italian.....	Machineman.....	29
	5 Joseph Sandona.....	do.....	Machine helper....	28
	9 Ed More.....	Belgian.....	do.....	26
	9 E. Johnson.....	American.....	Mucker.....	21
	9 R. A. Barnes.....	do.....	Machine helper....	50
	11 Ray Gossett.....	do.....	do.....	32
	11 John Variamos.....	Greek.....	Bulldozer.....	35
Feb.	31 John Bogdanovich.....	Austrian.....	Mucker.....	42
	2 Mike Cook.....	Russian.....	do.....	40
	4 Louis Welvert.....	Belgian.....	Steel packer.....	23
	8 M. H. Gedney.....	American.....	Foreman.....	60
	13 James Zeoff.....	Russian.....	Mucker.....	30
	16 Mike Zugic.....	Montenegrin.....	Machineman.....	24
	18 Emelio Petrasso.....	Italian.....	Machine helper....	26
	26 Pietro Bonna.....	do.....	Laborer.....	40
	29 Mike Chalovich...?	Montenegrin.....	Machine helper....	35
	Mar. 3 L. P. Hamilton.....	American.....	do.....	35
	6 Math Jakominich.....		Machine helper....	25
	13 Mike Knaz.....	Montenegrin.....	do.....	38
	14 Steven Rolando.....	Italian.....	Mucker.....	36
	16 Gus Erickson.....	Swede.....	Contractor.....	28
	16 Andrew Erickson.....	do.....	do.....	22
	19 John Jarico.....	Italian.....	Machine helper....	40
	19 Kosta Guisino.....	Montenegrin.....	Chute puncher....	37
	25 Dan Sakatos.....	Greek.....	Machineman.....	30
	Apr. 6 Louis Odolovich.....	Austrian.....	Mucker.....	45
	10 J. M. Steele.....	American.....	Miner.....	45
	18 Costa Caramouzie.....	Greek.....	Mucker.....	45
May	18 James Bowie.....	Scotch.....	Machineman.....	43
	21 Peter E. Genatos.....	Greek.....	Mucker.....	28
	24 Antonio Dalla.....	Italian.....	Bulldozer.....	31
	29 Ralph Marcine.....	do.....	Timberman.....	25
	2 Nick Demer.....	Greek.....	Bulldozer.....	25
	4 George T. Quinge.....	Norwegian.....	Carpenter.....	45
	5 Mike Moras.....	Montenegrin.....	Bulldozer.....	36

mines in Alaska during the year ended Dec. 31, 1916.

Name of company and mine.	Nature and cause of accident.
Alaska Gastineau Mining Co.; Perseverance.	Waste on which machine was set moved, and machine fell on operator, breaking both his legs.
do.	Injured was standing with hand resting on machine when rock fell from roof, bruising and lacerating hand.
do.	First joint, index finger, right hand, badly crushed by machine which rolled over while injured was lifting it.
Jualin Alaska Mines Co.	Crushed ends of fourth and fifth fingers between a piece of ore and car. The ore protruded over top of car and caught on timber.
Alaska Gastineau Mining Co.; Perseverance.	Right foot bruised by fall of rock while carrying drill steel through stope.
Alaska Gold-Belt Co.	Collar bone broken. Clothing caught in gear of diamond drill and injured was thrown against chuck nut.
Alaska Gastineau Mining Co.; Perseverance.	Rush of ore in chute while barring down. Fractured skull and badly cut and bruised shoulders.
Alaska Juneau Gold Mining Co.	Two ribs broken on left side by rock falling from chute while barring down.
Alaska Gastineau Mining Co.; Perseverance.	Bone in right hand broken by fall of rock in drift.
Alaska Mexican Gold Mining Co.; Mexican.	Injured was carrying drill steel across stope and fell; steel bruised the third finger on right hand.
Ellamar Mining Co.; Ellamar.	Leg broken by fall of rock in waste raise.
Alaska Gastineau Mining Co.; Perseverance.	Thumb of right hand crushed and hand lacerated. Injured attempted to hold coupling link between two cars with his candlestick, when motor bumped against them and caught his hand between the bumpers.
do.	Small piece of steel from drill cut tissue of left eye.
Alaska Treadwell Gold Mining Co.; Treadwell.	Injured was climbing ladder when his partner dropped monkey wrench, which fell on the middle finger of his right hand.
Alaska United Gold Mining Co.	Tip of thumb on right hand crushed by a rock which slipped while injured was unloading car.
Alaska Gastineau Mining Co.; Perseverance.	Eyes filled with dirt from small explosion caused by drilling into cut-off hole.
do.	While watching car being loaded at chute, rock bounced off car and bruised left foot.
do.	Jack bar slipped from column which injured was tightening and threw him down, causing a compound fracture of first phalanx of index finger.
do.	Injured was carrying machine over waste pile when he fell with machine on top of him, causing a right inguinal hernia. (Old case.)
Alaska Juneau Gold Mining Co.	Ruptured by strain while rolling a heavy rock.
Alaska Gastineau Mining Co.; Perseverance.	Contused and punctured wounds of right arm, chest, both hands, and upper left leg, due to flying particles from explosion caused by drilling into missed hole.
do.	Contused wound of left wrist due to flying particles from explosion caused by drilling into missed hole.
do.	Injured was helping lower machine when it fell off the arm, dropping on his right foot and bruising it severely.
do.	Injured had been barring down in a chute. He laid bar on plank and rock fell and struck one end of the bar, throwing the other against his face, lacerating the lip and eye.
do.	Injured was standing with his left hand on crossbar when rock fell from roof and crushed fourth finger between rock and bar.
Alaska Juneau Gold Mining Co.	End of left index finger broken by fall of rock.
Kennecott Copper Corporation.	Fell through raise from one stope to another while trying to start ore. General contusions, especially about right arm and shoulder.
Alaska Gastineau Mining Co.; Perseverance.	Clearing tracks at chute while train was loading; did not hear "go ahead" signal and was struck by car when train started.
Alaska United Gold Mining Co.; Ready Bullion.	Lacerated wound of left eye, fracture of left radius, and fracture of second rib, both sides.
Alaska Gastineau Mining Co.; Perseverance.	Climbing up raise on rope, slipped and fell. Cut head and sprained ankle.
Alaska Treadwell Gold Mining Co.; Treadwell.	Third rib fractured while coupling cars.
Alaska Gastineau Mining Co.; Perseverance.	While preparing to bulldoze rock in stope, a piece of ore rolled down pile, breaking left leg and one rib.
do.	Contusion of right foot, fracture of fourth phalanx. Injured was hoisting planks in chute when one fell and struck his foot.
do.	Compound fracture of inferior maxillary and lacerated wound of neck. Injured was barring down at chute when rock fell on one end of bar, which flew up and struck his jaw.
Alaska Juneau Gold Mining Co.	Right hip and groin bruised. Injured was working on construction of tramway when rock rolled down the hill and struck him.
Alaska Gastineau Mining Co.; Perseverance.	Fracture of inferior maxillary, lacerated wound of right cheek and scalp. Injured was barring down at chute and had his bar over check board when ore fell on bar, which struck his jaw and knocked him off platform.

TABLE 5.—*Serious accidents underground in lode mines in*

Date of accident.	Name of person.	Nationality.	Occupation.	Age.
May 8	Otto Hocking.....	Swede.....	Chute puncher.....	35
8	Thomas Griffen.....	Irish.....	Mucker.....	26
9	Rista W. Lurich.....	Servian.....	Machine helper.....	26
13	Joseph Rodich.....	Austrian.....	Machineman.....	26
22	Thomas Basoff.....	Russian.....	Chute puncher.....	32
22	Emil Kalmutsky.....	do.....	Machineman.....	25
26	Jacob Nordahl.....	Norwegian.....	Pipeman.....	43
26	Albert Henden.....	do.....	Machineman.....	30
26	Mike Ogden.....	Austrian.....	Chute puncher.....	21
30	Algot Johnson.....	Swede.....	Mucker.....	21
June 8	John Wilson.....	Finlander.....	Machineman.....	27
10	Antonio Bertolini.....	Tyrollian.....	Laborer.....	40
13	Peter Perovich.....	Montenegrin.....	Machineman's helper.....	39
15	C. Garavatti.....	Italian.....	Steel nipper.....	26
16	Gust E. Johnson.....	Swede.....	Rock breaker and driver.....	18
17	C. W. Carlson.....	American.....	Miner.....	34
27	Peter Deibrich.....	Montenegrin.....	Chute puncher.....	34
29	J. Flenor.....	American.....	Motorman.....	45
July 3	Samuel Aboff.....	Russian.....	Machineman's helper.....	22
7	Robert Rosandich.....	Austrian.....	Laborer.....	25
10	Nick Pope.....	do.....	Machine man's helper.....	40
20	Gus James.....	Greek.....	Mucker.....	27
22	Vuko Tomasovich.....	Montenegrin.....	do.....	23
23	Peter M. Nikaloff.....	Belgian.....	Chute puncher's helper.....	39
25	Obren Odalovich.....	Austrian.....	Chute puncher.....	24
25	Henry Jackson.....	Finlander.....	Machineman.....	38
27	Christ Nick.....	Turk.....	Laborer.....	39
29	James Jarden.....	Austrian.....	Cage man.....	29
Aug. 4	L. Watt.....	Scotchman.....	Machineman.....	38
8	George Sabolek.....	Austrian.....	Hand miner.....	37
12	Nick Kupoff.....	Russian.....	Chute puncher.....	25
12	Mike Voinovich.....	Montenegrin.....	Laborer.....	29
15	Frank Powers.....	American.....	do.....	20
20	Mike Odal.....	Austrian.....	Machine man.....	22
25	Alex Sagoff.....	Russian.....	Car man.....	23
28	Samuel Karadish.....	Montenegrin.....	Machine man.....	25

Alaska during the year ended Dec. 31, 1916—Continued.

Name of company and mine.	Nature and cause of accident.
Alaska Gastineau Mining Co.; Perseverance.	Lacerated wound of left hand, compound fractures of third and fourth fingers. Injured was standing with hand on check board when rock fell on it.
Alaska Juneau Gold Mining Co.....	Index finger of right hand cut and bruised. Barring down in tunnel when rock fell from roof.
.....do.....	Instep bruised by piece of drill steel falling down manway.
Alaska Gastineau Mining Co.; Perseverance.	Ulcer on eyeball due to cut from flying piece of steel from drill while starting hole.
.....do.....	Contusion of first toe, left foot. Injured was standing before chute when piece of ore fell over chute door and struck his toe.
Alaska Juneau Gold Mining Co.....	Instep bruised by machine which fell on foot.
.....do.....	Right thigh cut by rock blown out of air pipe while Nordahl was cleaning the line.
.....do.....	Face and nose cut and bruised. Henden was assisting Nordahl in removing rock and dirt from the air pipe.
Goodro Mining Co.; May Group.....	Rupture due to lifting excessive weight.
Kennecott Copper Corp.; Bonanza.....	End of second toe, left foot, crushed by rock falling from chute.
Jualin Alaska Mines Co.; Jualin.....	First finger of right hand mashed. Rock projecting from car which injured was pushing struck timbers, catching his finger between the rock and the car.
Alaska Treadwell Gold Mining Co.; Treadwell.	Head and face cut and right hip injured by falling rock in stope.
Alaska Gastineau Mining Co.; Perseverance.	Lacerated wound of left eye and contusion of left ankle. Injured was walking across stope when the broken ore moved, throwing him down and bruising him.
Alaska United Gold Mining Co.; 700-Foot.	Left leg fractured by fall of rock from roof of stope.
Alaska United Gold Mining Co.; Ready Bullion.	Collar bone broken. Injured was caught between car and chute.
Alaska Free Gold Co.....	Slipped on ice in raise and fell, dislocating shoulder.
Alaska Gastineau Mining Co.; Perseverance.	Compound fracture of fourth finger of right hand. Rock fell on bar and caught finger between bar and chute board.
.....do.....	Contused wounds of left ankle. While walking along drift, turned to look at a chute and ran into a truck.
.....do.....	Back, legs, and shoulder injured by fall of rock in stope.
Alaska Juneau Gold Mining Co.....	Back, shoulder, and left side bruised. Injured and two others were pushing loaded dump car out of tunnel when motor train came around curve and struck dump car, throwing it off track and catching injured between dump car and side of tunnel.
Alaska Gastineau Mining Co.; Perseverance.	Face and hands burned by flame from acetylene gas generated when injured poured carbide into can containing water.
Ellamar Mining Co.; Ellamar.....	Right leg bruised and thigh fractured by falling in a chute.
Alaska Gastineau Mining Co.; Perseverance.	Fractured left tibia when hit by car dump which injured had just unhooked.
.....do.....	Contusion of hips. Stepped between car and chute just as train started.
.....do.....	Compound fracture of nose and contused wound of scalp. Injured had bar over chute board when rock fell on inside end of bar, throwing bar against his face and knocking him off the platform into the car.
Alaska Juneau Gold Mining Co.....	Right eye bruised—vision impaired—by small rock falling while ascending ladder in raise.
.....do.....	Right foot crushed. Injured was removing rocks from side of drift when ore train came up behind him. His foot jammed between a car and a piece of rock.
Alaska Gastineau Mining Co.; Perseverance.	Compound fracture of fifth metacarpal bone, right hand. Injured was raising oil tank with winch when handle broke and flew back, striking him on the hand.
.....do.....	Index finger crushed; amputation necessary. Injured was walking with bar on his shoulder when he fell, catching his hand between the bar and a rock.
Alaska Mexican Gold Mining Co.; Mexican.	Right arm crushed; amputated below elbow. Injured went up chute to determine position to place powder to start chute when rock fell on him.
Alaska Gastineau Mining Co.; Perseverance.	Fracture of first toe, left foot. Rock rolled over chute board and fell on toe.
Alaska Juneau Gold Mining Co.....	Face and eyes cut by flying rocks from explosion of a cap or piece of dynamite in the waste injured was shoveling.
.....do.....	Third finger, left hand, badly bruised while loading steel beams on truck.
Alaska Gastineau Mining Co.; Perseverance.	Fracture of twelfth dorsal vertebra and contused wounds of scalp and face. Fall of rock in a stope.
Kennecott Copper Corp.; Bonanza.....	End of third finger, left hand, lacerated by falling rock which caught injured's hand between car and bar.
Alaska Gastineau Mining Co.; Perseverance.	Contused wounds of third and fourth fingers, left hand. Injured was pulling up steel and caught hand under cable.

TABLE 5.—*Serious accidents underground in lode mines in*

Date of accident.	Name of person.	Nationality.	Occupation.	Age.
Sept.	9 Oscar Bondeson.....	Swede.....	Mucker.....	28
	9 Nunsio Greko.....	Italian.....	Laborer.....	30
	9 Alex Govadenovich.....	Montenegrin.....	do.....	28
	16 Attilio Costa.....	Italian.....	Machine man.....	32
	16 John Yakich.....	Montenegrin.....	Mucker.....	33
	22 John B. Larson.....	Finlander.....	Timberman and hoistman.....	34
	23 Lawrence Linden.....	American.....	Miner.....	40
Oct.	29 Battista Alilo.....	Italian.....	Machine man.....	26
	2 George Besoloff.....	Russian.....	Mucker.....	40
	2 Charles Penn.....	Austrian.....	Machine man.....	29
	3 E. Patzold.....	German.....	Miner.....	50
	3 G. Tomma.....	Belgian.....	do.....	34
	5 Thomas Tsolokas.....	Greek.....	Bulldozer.....	31
	15 Eli Lelich.....	Austrian.....	Machine man's helper.....	26
	15 Mike Jelich.....	do.....	Bulldozer.....	45
	15 George Gogoff.....	Russian.....	Chute puncher.....	21
	19 Oscar Ness.....	Finlander.....	Machine man.....	29
	22 Red Alechsich.....	Montenegrin.....	do.....	34
Nov.	3 Charles Penn.....	Austrian.....	do.....	29
	11 Earl Rhodes.....	American.....	Machine man's helper.....	23
	13 W. W. Eggen.....	do.....	Motorman.....	22
	27 Knut Danielson.....	Swede.....	Machine man.....	37
Dec.	27 Peter Miller.....	Austrian.....	do.....	40
	1 Peter Batello.....	Italian.....	do.....	42
	2 Frank Peters.....	Indian.....	Mucker.....	26
	7 Lazo Kovack.....	Austrian.....	Chute puncher.....	52
	8 Ely Radovich.....	Montenegrin.....	Contractor.....	30
	9 Ire Hall.....	Finlander.....	Mucker.....	27
	31 Herman Lurda.....	do.....	Machine man.....	40

Alaska during the year ended Dec. 31, 1916—Continued.

Name of company and mine.	Nature and cause of accident.
Chicagoff Mining Co.; Chicagoff	Leg broken by fall of rock in drift.
Alaska Juneau Gold Mining Co.	Head, face, and hip bruised. Overcome by powder gas and fell down chute.
do.	Top of head cut, shoulders, back, and upper part of legs bruised.
do.	While injured was helping put skip on track the steel bar that he was using lost its hold and he fell down an incline raise about 125 feet.
Alaska Gastineau Mining Co.; Perseverance.	Lacerated wounds of second and third fingers, right hand.
do.	Column bar slipped while injured was adjusting head block and caught his fingers between bar and wall.
Alaska Juneau Gold Mining Co.	Contused wound of left foot caused by injured's dropping rock on foot.
do.	Two or three ribs broken on left side. Injured was standing on carpenter's horse setting hangers for trolley wire when the horse turned over and threw him to the ground.
Kennecott Copper Corp.; Bonanza.	Two cuts on head. Rock rolled down from pile of ore thought to be frozen.
Alaska Treadwell Gold Mining Co.; Treadwell.	Left arm fractured by a piece of steel which fell down a raise while injured was setting timbers.
Chicagoff Mining Co.; Chicagoff	Lower leg broken. Injured was riding timber truck and attempted to stop same with foot.
Alaska Gastineau Mining Co.	Fracture of fourth metacarpal bone, right hand. Injured was pulling on jack bar when it slipped and caused him to fall on his hand.
Jualin Alaska Mines Co.	Leg bruised by fall of rock in stope.
do.	Flesh torn from leg by hook on car. Injured was walking along drift and did not see car in time to get out of way.
Alaska Gastineau Mining Co.; Perseverance.	Contused wounds of right forearm from fall of rock in chute.
do.	Contused wounds of first, second, and third fingers, left hand.
do.	Injured tried to start small skip stuck in a raise. It gave way suddenly and his hand caught between the skip and timbers.
do.	Back hurt and internal injuries; fell from ladder.
do.	Lacerated wound of index finger, right hand. Finger caught between bar and rock.
Alaska Juneau Gold Mining Co.	Left eye injured by piece of steel flying from the bit of a near-by machine.
do.	Left foot bruised by falling jack bar which injured was using in setting drift timbers.
Alaska Gastineau Mining Co.; Perseverance.	Face and head cut, muscles of neck and back sprained. Injured was thrown down by an air blast.
do.	Felon on right index finger due to infection of wound from a silver on a drill rod.
do.	Left leg bruised and cut below knee. Injured ran motor into standing trip and caught leg between a car and the motor.
Alaska United Gold Mining Co.; 700-Foot.	Left leg broken below knee. Had helper loosen clamp on tripod and machine fell off, striking injured on leg.
Jualin Alaska Mines Co.	Bruised leg. Fall of ground in a raise.
do.	Bruised finger. Steel in machine broke, throwing it against wall.
Alaska Juneau Gold Mining Co.	One or two small bones broken in foot. Injured was lifting rock into the car when it broke and fell on foot.
Alaska Gastineau Mining Co.; Perseverance.	Right side and both arms injured. Injured stepped off motor while in motion and grabbed the ladder, and the moving car crushed him between car and ladder.
Ellamar Mining Co.; Ellamar.	Burns on face and hands, bruises on arms, legs, and body.
do.	Blown down a raise by an explosion of methane.
Alaska Juneau Gold Mining Co.	Ulcer on cornea, right eye, due to bruise from flying rock.
Alaska Gastineau Mining Co.; Perseverance.	Back and right leg bruised. Motor ran into injured while standing in drift so smoky that he did not see motor approaching.

TABLE 6.—*Serious accidents in placer mines*

Date of accident.	Name of person.	Nationality.	Occupation.	Age.
Apr. 1	Axel Sandberg.....		Operator.....	
5	Paul Schultz.....		do.....	
18	James O'Dea.....		Machinist.....	
19	James Mulvehill.....		Laborer.....	45
25	William Kelly.....		Operator.....	
25	Samuel Lowery.....		Laborer.....	
..	William Flater.....			
May 11	Adolph Holtman.....		Mucker.....	
19	Thomas Isaacson.....	Finlander.....	Miner.....	46
June 1	George Getton.....	Russian.....	do.....	
..	E. Richards.....	English.....	Carpenter.....	42
July 10	George McFeely.....	Scotch.....	Timberman.....	
14	Harvey G Dorr.....	German.....	Mucker.....	
29	John Neilson.....		Laborer.....	33
Aug. 4	Daniel Dooling.....	American.....	Operator.....	44
9	Carl Karleen.....		Engineer.....	
10	Alex Kannoff.....	Russian.....	Miner.....	
18	Abraham Jukko.....	Finlander.....	do.....	
..	O. W. Flowers.....		Operator.....	
Sept. 13	Louis Peterson.....		Pipe man.....	
Oct. 5	Dommick Brondino.....	Italian.....	Miner.....	34

in Alaska during the year ended Dec. 31, 1916.

Name of operator and mine.	Nature and cause of accident.
Kleinschmidt & Sandberg; Niggerhead Association.	Injured lost sight of 1 eye and injured the other, lost 4 fingers on right hand, had right leg broken in 2 places, and hearing affected by an explosion of dynamite and caps. He was alone at the time and did not know the cause of the explosion.
Paul Schnitz.....	Burned and bruised by explosion caused by drilling into missed hole.
Yukon Gold Co.....	Accidentally struck on the head by a hammer which caused a slight wound. This later became infected.
.....do.....	While thawing frozen ground ahead of a dredge, injured stepped near live steam point and crust gave way, plunging his foot into hot muck and scalding his leg above the shoe top.
Kelly, Sampson & Colbert; Gold Dollar Association.	Kelly raised lighted candle to examine roof and ignited gas pocket which burned his face and hands.
.....do.....	Lowery was with Kelly. Face and hands were burned.
Cleveland & Howell; Mohawk.....	Injured was timbering drift when the gravel caved from the side and broke his leg below the knee.
Finley & Paterson; Deep Channel Association.	Leg caught between 2 large pieces of frozen gravel and crushed so badly that amputation at thigh was necessary.
Howell & Cleveland; Mohawk and Lorraine claims.	Leg broken by slab of bedrock which slid out of the side of the drift.
.....do.....	Eye injured by piece of flying gravel.
Flume Dredge Co.; No. 1, above Melsing Creek.	While injured was assisting in unloading a line shaft from a wagon it fell and broke his leg.
August Peterson.....	Injured was timbering shaft when he lost his balance and fell 25 feet, straining his back.
Frank Waskey; No. 1, below Willow Creek.	Leg broken by a bowlder which fell off a conveyor in a hydraulic pipe.
Yukon Gold Co.....	Injured, while thawing frozen ground ahead of dredge, attempted to remove plank from steam-point hole and slipped, plunging his legs into hole and scalding them.
Doelling, McIntosh & Patterson; No. 1, below Myrtle Creek.	Bowlder fell from face of drift and struck injured on foot, breaking several small bones.
Julian Dredge Co.....	Injured stood on winch drum on dredge without notifying winchman, who started winch and threw injured against the clutch, lacerating thigh and calf of right leg.
Howell & Cleveland; Mohawk & Lorraine.	Injured entered old workings and ignited a pocket of gas, which burned his face and hands.
.....do.....	Do.
O. W. Flowers's dredge.....	Injured thrust his hand through spokes of gear wheel to screw down grease cup when winchman, who had not been notified that anyone was working on engine, started the winch, crushing the injured's arm so badly that amputation was necessary.
Pioneer Mining Co.; Ault.....	Injured lost control of a nozzle which swung around and broke 2 of his ribs.
Howell & Cleveland; Mohawk & Lorraine.	Eye was struck by a chip from a bowlder which he struck with his pick.

TABLE 7.—*Serious accidents in*

Dredging.						Hydraulic mining.							
By machinery.	By electricity.	By boiler explosion or bursting steam pipes.	By falls of persons.	By tools.	By other causes.	By cave of bank.	By explosives.	By hydraulic giants.	By falls of persons.	By rock while handling.	By tools.	By machinery, derricks, etc.	By other causes.
1	2	3	4	5	6	7	8	9	10	11	12	13	14
				1	2								
					1			1					
1													
					1								
										1			
1													
2				1	4			1		1			

placer mines in Alaska, by causes, 1916.

Underground.									Result of injury. (Time lost more than 14 days.)			Name of company.
By fall of roof or gravel.	By timber or hand tools.	By explosives.	By falls of persons.	By haulage.	By drilling.	By hoisting.	By steam (when steam points are used).	By other causes.	Permanent total disability.	Permanent partial disability.	Temporary d is- ability.	
15	16	17	18	19	20	21	22	23				
	1					1				1	2	Driscoll Bros.
		1										Kleinschmidt & Sandberg.
		1										Paul Schultz.
1												3 Yukon Gold Co.
												1 Dooling & Paterson.
										1		1 Pioneer Mining Co.
												Julian Dredge Co.
		2									1	1 Kelly, Sampson & Colbert.
1										1		2 Finley & Paterson.
2								4				6 Howell & Cleveland.
			1									1 Flume Dredge Co.
												1 August Peterson.
												1 Frank Waskey.
												1 O. W. Flowers Dredge.
4	1	4	1			1		4		3	21	

DATA ON DREDGES, LODGE MINES, AND PLACER MINES.

Data on dredges, lodge mines, and placer mines in Alaska for the year 1916 are presented in tables 8 to 10. Table 8 gives important details in regard to dredges. Table 9 gives the names of mines, operator, and manager, with post office address, for the lodge mines of the Territory, by districts. Table 10 gives the names of placer claims and the creek on which the claim is situated, with name and address of operator, by districts.

TABLE 8.—Data on

Name.	Alaska address.	Creek or river.	Type.	Size of bucket.	Bucket line.
<i>Seward Peninsula.</i>					
American Gold Dredging Co.	York.	Anicovik.	Flume.	Cu. ft. 2	Open
Do.	do.	do.	do.	1½	Close.
American Tin Dredging Co.	do.	Buck.			
Arctic Dredging Co.	Nome.	Hobson.	Belt stacker.	2½	Open.
Arctic Creek Dredge.	do.	Arctic.			
Bangor Creek Dredging Co.	do.	Bangor.	Belt stacker.	3½	Open.
Bering Dredging Co.	Taylor.	Kougarok.			
Blue Goose Mining Co.	Council.	Ophir.	Belt stacker.		Close.
Candle Creek Dredging Co.	Candle.	Candle.	Flume.	1½	Open.
Center Creek Dredging Co.	Nome.	Center.			
Ernst-Alaska Dredging Co.	do.	Nome Beach.	Flume.	1½	Open.
Flowers Dredge.	Solomon.	Solomon.	B u c k e t stacker.	2½	do.
Flume Dredge Co.	Council.	Melting.	Flume.	2½	do.
Do.	do.	do.	do.	2½	do.
Fries Dredging Co.	Deering.	Inmachuk.			
Glacier Creek Dredge.	Nome.	Glacier.			
Goose Creek Dredge.	Dickson.	Goose.			
Hastings Creek Dredge.	Nome.	Hastings.			
Inmachuk Dredging Co.*	Candle.	Inmachuk.	Flume.	3	Open.
Johnson Dredge.	do.	Kugruk.	do.	3	Close.
Julian Mining Co.	Nome.	Osborne.	Belt stacker.	2½	Open.
Kelliher Dredge.	Taylor.	Kougarok.	B u c k e t stacker.	2½	do.
Kimball Co., C. E.	Nome.	Solomon.	Flume.	2½	do.
Do.	do.	do.	do.	2½	do.
Moody Mining Co.	do.	Canyon.	do.	2½	do.
Northern Light Mining Co.	Council.	Ophir.	do.	2½	do.
Nome Consolidated Dredging Co.	Nome.	Burbon.	Belt stacker.	7	Close.
Do.	do.	Wonder.	do.	7	Open.
Do.	do.	do.	do.	5	do.
Nome-Montana-New Mexico Mining & Dredging Consolidated.*	Solomon.	Solomon.	B u c k e t stacker.	7	do.
Oro Dredging Co.	Council.	Elkhorn.	Flume.	1½	do.
Plain Mining & Dredging Co.*	Nome.	Nome.	B u c k e t stacker.	2½	do.
Ruby Dredging Co.	do.	Casadepega.	Flume.	2½	do.
Selverson & Johnson.	Dickson.	Solomon.	B u c k e t stacker.	2½	do.
Seward Dredging Co.	do.	do.	Belt stacker.	5	Close.
Shovel Creek Gold Dredging Co.*	do.	Shovel.	do.	2½	do.
Solomon Dredging Co.*	do.	Solomon.	do.	3½	do.
Sunset Mining Co.	Teller.	Sunset.			
Uplift Mining Co.	Council.	Camp.			
Warm Creek Dredging Co.	do.	Warm.	Flume.	2½	Open.
Wild Goose Mining & Trading Co.	Nome.	Ophir.	Belt stacker.	3½	Close.
Willow Creek Dredging Co.*	Dickson.	Willow.	do.	3	Open.
Windy Creek Dredge.	Teller.	Windy.			
York Tin Dredging Co.	York.	Grouse.	Flume.	2½	Open.
<i>Cache Creek.</i>					
Cache Creek Dredging Co.	Cache via Susitna.	Cache.	do.	7	Close.
<i>Circle.</i>					
Berry Dredging Co., C. J.	Circle.	Mastodon.			
<i>Fairbanks.</i>					
Fairbanks Gold Mining Co.	Fairbanks.	Fairbanks.	B u c k e t stacker.	3½	Open.
<i>Iditarod.</i>					
Otter Creek Dredging Co.	Iditarod.	Otter.	Flume.	3½	Close.
Do.	do.	Black.	do.	2½	do.
Yukon Gold Co.	do.	Flat.	Belt stacker.	7½	do.
<i>Kenai Peninsula.</i>					
Herron Dredging Co.*	Hope.	6-Mile.			
<i>Ruby.</i>					
Yukon Gold Co.	Ruby.	Greenstone.			

* Not operating in 1916.

DREDGES.

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dredges in Alaska, 1916.

Maximum digging depth.	Rated horsepower.	Source of power.	Actual capacity in yards per 24 hours.	Dimensions of hull.	Manager.
Feet.				Feet.	
12	80	Distillate.....			B. Bernard.
15	50	do.....	1,200	28 by 54	Do.
14	92	Distillate.....	1,240	30 by 60	Nels Nelson.
35	140	Crude oil.....	2,000	36 by 92	F. Middaugh.
17	90	Steam.....	1,600	32 by 96	C. Servatius.
14	50	Distillate.....	1,000	24 by 40	C. Mitchell.
12	40	Distillate.....	700	24 by 46	John Mathews.
12	80	Steam.....	1,000	34 by 68	A. N. Kittilsen.
18	50	Distillate.....	1,000	26 by 50	Frank Sundquist.
12	60	do.....	1,000	28 by 60	Andy Anderson.
					Phillip Ernst.
					O. W. Flowers.
					C. E. Kimball.
					Do.
					H. Fries.
					H. Ames & A. Guinan.
					George A. Adams.
					Joseph Bellevue.
18	107	Distillate.....	1,400	30 by 62	Iver Johnson.
16	107	do.....	2,000	30 by 62	V. Julian.
15	87	do.....	1,000	30 by 60	James Kelliber.
15	90	do.....	1,000	34 by 60	
13	60	do.....	1,000	30 by 60	C. E. Kimball.
13	60	do.....	1,000	28 by 60	Do.
12	60	do.....	1,000	28 by 60	C. L. Peck.
12	50	do.....	1,000	28 by 60	Gilbert A. Russell.
35	310	Electricity.....	7,000	48 by 114	John Miles.
70	330	do.....	6,000	48 by 114	Do.
70	330	do.....	6,000	48 by 114	Do.
16	120	Steam.....	1,600	38 by 87	G. F. Ramsey.
8	18	Distillate.....		16 by 24	Charles Spencer.
12	60	Steam.....	1,000	34 by 68	J. F. Plein.
9	90	Distillate.....	1,200	30 by 60	W. W. Johnson.
12	80	Steam.....	1,000	34 by 60	C. O. Seiverson.
12	225	Electricity.....	4,000	40 by 86	R. S. Ogilby.
15	120	Distillate.....	1,200		Corey C. Brayton.
25	130	Steam.....	2,000	45 by 85	J. A. Malloch.
					Max Hirsberg.
12	75	Distillate.....	1,000	25½ by 60	A. N. Kittilsen.
25	160	do.....	2,200	30 by 75	Charles Milosek.
12	90	do.....	1,000	40 by 60	Fred M. Ayer.
					Jerry L. Wilson.
					J. A. Welch.
14	87	Distillate.....	1,000	28 by 60	W. W. Johnson.
30	295	Steam.....	2,000	64 by 87	Ed. L. Smith.
					C. J. Berry
16	85	Steam.....	1,000	45 by 90	G. Aarons.
18	107	Distillate.....	2,000	30 by 62½	J. E. Riley.
26	300	Electricity.....	5,000	46½ by 95	Do.
					E. A. Austin.
					Charles Herron.

TABLE 9.—Data on lode mines in Alaska, 1916.

SOUTHEASTERN ALASKA.

Operator.	Mine.	Local address.	Manager.
Admiralty Alaska Gold Mining Co.		Juneau	W. S. Peckovich.
Alaska Gold Belt Mining Co.		do.	A. B. Dodd.
Alaska Gold Mines Co.	Perseverance.	do.	B. L. Thane.
Alaska Industrial Co. (See Sulzer, C. A.).			
Alaska Juneau Gold Mining Co.	Alaska Juneau.	Juneau	P. R. Bradley.
Alaska Mexican Gold Mining Co.	Mexican.	Treadwell.	Do.
Alaska Treadwell Gold Mining Co.	Treadwell.	do.	Do.
Alaska United Gold Mining Co.	700 Foot.	do.	Do.
Do.	Ready Bullion.	do.	Do.
Chicago Mining Co.	Chicago.	Chichagof.	J. R. Freeburn.
Crystal Gold Mining Co.		Snettlaham.	Bernard Heine.
Dunton Gold Mining Co.	Dunton.	Hollis.	C. H. Dunton.
Eagle River Mining Co.	Eagle River.	Amalga.	B. L. Thane.
Ebner Gold Mining Co.	Ebner.	Juneau.	George Osborne.
Goldstream Mining Co.	Goldstream.	Ketchikan.	
Goodro Mining Co.	Goodro.	do.	S. J. Goodro.
Granby Con. Ming., Smelting, & Power Co.	It.	Hadley.	M. W. Sweetser.
Do.			Do.
Jualin Alaska Mines Co.	Mamie.	Juneau.	H. G. Young.
Juneau Sea Level copper mines.	Jualin.	Chichagof.	E. E. Fleming.
Kensington Mining Co.	Kensington.	do.	B. L. Thane.
Lakina & Tagish Mines Co.	Cymru-Moira.	Ketchikan.	J. L. Harper.
Mount Andrew Mining Co.	Mount Andrew.	do.	W. J. Rogers.
Nestor Mining Co.	Nestor.	Hadley.	John Wick.
Northland Development Co.		Crags.	P. A. Tucker.
Pacific Coast Gypsum Co.	Gypsum.	Gypsum.	D. C. Stapleton.
Princeton Mining & Milling Co.		Dolomi.	B. A. Keadley.
Ready Bullion Mining Co.	Ready Bullion.	Hollis.	H. W. Webber.
Rush & Brown.		Kasaan.	U. S. Rush.
Sea Level mine.	Sea Level.	Ketchikan.	
Sulzer, Chas. A. (lessee Alaska Industrial Co. property).	Jumbo.	Sulzer.	Chas. A. Sulzer.
Thane Exploration Co., B. L.		Juneau.	B. L. Thane.

COPPER RIVER DISTRICT.

Alaska Consolidated Copper Co.	Nugget Creek.	Strelina.	H. W. DuBois.
Great Northern Development Co.	Gray's Copper Mountain.	Phillips.	E. F. Gray.
Hubbard-Elliott Copper Co.	Hubbard-Elliott.	Elliott Creek via Strelina.	A. J. Elliott.
Kennecott Copper Corporation.	Bonanza.	Kennecott.	E. T. Stannard.
Do.	Jumbo.	do.	Do.
Lakina & Tagish Mines Co.			J. L. Harper.
Mother Lode Copper Mines Co.	Mother Lode.	McCarthy.	W. B. Handcock.
North Midas Copper Co.		Strelina.	O. J. Berg.

PRINCE WILLIAM SOUND.

Alaska Gold Mining Co. (formerly Black Diamond).		Valdez.	Geo. F. White.
Alaska Mines Corporation.	Schlosser.	do.	E. D. Reiter.
Alice Mines (Ltd.).	Alice.	do.	M. J. Callaghan.
Bennett-Daley mine.	Bennett-Daley.	do.	Samuel Pepper.
Big Four mine.	Big Four.	do.	A. Wilcox.
Black Diamond. (See Alaska Gold Mining Co.)			
Cameron-Johnson Gold Mining Co. (See Valdez Gold Co.)			
Cliff mine.	Cliff.	Valdez.	H. E. Ellis.
Cube Mines Co., The.		do.	Jefferson Divinney.
Ellamar Mining Co.	Ellamar.	Ellamar.	L. L. Middelkamp.
Ellis Imperial Mining Co.		Valdez.	H. E. Ellis.
Fidalgo Mining Co.	Fidalgo.	Ellamar.	William Mackintosh.
Galena Bay Mining Co.	Galena Bay.	Valdez.	Chas. Simonstead.
Gold King mine.	Gold King.	do.	Owners: Frank Gustavson, Angus Chishom, Gus Nelson.
Granby Consolidated Mining, Smelting & Power Co.	Midas.	do.	Palmer J. Cook.
Granite Gold Mining Co.	Granite.	do.	W. R. Millard.
Irish Cove Copper Co.	Irish Cove.	Ellamar.	W. A. Dickey.
Kennecott Copper Corporation.	Beatson.	Latouche.	E. T. Stannard.
Do.	Bonanza.	do.	Do.
Landlock Bay Copper Co.	Landlock Bay.	Ellamar.	W. A. Rystrom.

TABLE 9.—Data on lode mines in Alaska, 1916—Continued.

PRINCE WILLIAM SOUND—Continued.

Operator.	Mine.	Local address.	Manager.
Mineral King Mining Co.....	Mineral King.....	Valdez.....	Russel Herman and Glen Eaton.
Mountain King mine.....	Mountain King.....	do.....	W. L. Smith.
Ramsey-Rutherford Mining Co.....	Ramsey-Rutherford. ford.	do.....	Henry Deyo.
Reynolds-Alaska Development Co.....	Sea Coast.....	Latouche.....	Archie Hancock.
Sea Coast Mining Co.....	Sea Coast.....	Valdez.....	
Sealey-Davis Mining Co.....	Sealey-Davis.....	do.....	E. C. Sealey and J. M. Davis.
Sweepstake Mining Co.....	Sweepstake.....	do.....	A. L. Singletary.
Thomas-Cutross Mining Co.....	Bugaboo.....	do.....	Don M. Thomas.
Three Man Mining Co.....	Three Man.....	Ellamar.....	W. A. Dickie.
Valdez Gold Co. (formerly Cameron-John- son Mining Co.).	Valdez.....	Valdez.....	— Saint.
Valdez Mining Co.....	do.....	do.....	

KENAI PENINSULA.

Bluebell mine.....	Bluebell.....	Seward.....	Charles Hubbard.
John Gilpatrick.....	Gilpatrick.....	do.....	John Gilpatrick.
Hickey Mining Co.....	do.....	do.....	
Kenai-Alaska Gold Mining Co.....	do.....	do.....	J. R. Hyden.
Lucky Strike mine.....	Lucky Strike.....	do.....	John Hirshey.
Moose Pass Mining Co.....	Moose Pass.....	do.....	Henry Salisbury.
Primrose mine.....	Primrose.....	do.....	Charles Hubbard.
Ronan & James.....	do.....	do.....	
Slater, John B.....	do.....	do.....	John B. Slater.
Scheen-Lechner mine.....	Scheen-Lechner.....	do.....	H. Hoben.
Stetson Creek Mining Co.....	do.....	do.....	

WILLOW CREEK.

Alaska Free Gold Mining Co. (See Martin, Wm.).....	do.....	do.....	
Brooklyn Development Co.....	do.....	Knik.....	
Gold Bullion mine (see Willow Creek mine). Independence Gold Mines Co.....	Independence.....	Knik.....	L. S. Robe.
Mable Milling, Mining & Power Co.....	do.....	do.....	W. E. Bartholf.
Martin, William (lessee, Alaska Free Gold property).	do.....	do.....	William Martin.
Willow Creek mines (lessee, Gold Bullion property).	Gold Bullion.....	do.....	J. H. Collier.

FAIRBANKS DISTRICT.

American Eagle mine.....	American Eagle.....	Fairbanks.....	E. Tyndall.
Bondholder mine. The.....	Bondholder.....	do.....	St Scafford.
Chatham Mining Co.....	do.....	do.....	C. Crites.
Crites & Feldman.....	Crites & Feldman.....	do.....	Alois Friedrich.
Friedrich, Alois.....	do.....	do.....	L. G. Goyot.
Goyot, L. G.....	do.....	do.....	Joseph Henderson,
Homestake Mining Co.....	Homestake.....	do.....	lessee; G. St. George, owner.
Mayflower mine.....	Mayflower.....	do.....	Thos. Gilmour & Stevens.
Mispah mine.....	Mispah.....	do.....	A. Hess.
McArthur, W. T.....	Berry.....	do.....	W. T. McArthur.
McCarthy mine.....	McCarthy.....	Fairbanks.....	John McCarthy.
McNeill-Huddleson mine.....	McNeill-Huddleson.....	do.....	Mike McNeill.
Newsboy Mining Co.....	Newsboy.....	do.....	Louis Golden.
Poz & Contadi.....	Olnes.....	do.....	Joe Poz and John Contadi.
Reliance Mng. Co.....	Reliance.....	Fairbanks.....	— Spalding.
Rhoads-Hall Mining Co.....	Rhoads-Hall.....	do.....	L. B. Rhoads.
Tanana Hydraulic & Quartz Mng. Co.....	do.....	do.....	
Thomas, Hagel, McCann & Micheally.....	Berry.....	do.....	David Thomas, Frank Hagel, James McCann, and John Mi- cheally.
Wyoming & Colorado mine.....	Wyoming and Colorado.....	Cleary.....	Tony Goeseman.
Wood, Richard.....	Eagle Antimony..	Fairbanks.....	Richard Wood.

TABLE 10.—Data on placer mines in Alaska, 1916.

CHISANA DISTRICT.

Operator.	Claim.	Creek.	Local address.
Best & Green.....	No. 7.....	Bonanza.....	Chisana.
Hill & Jenney.....		Sargent.....	Do.
James & Nelson.....		Bonanza.....	Do.
McLellen & Fagelbury.....		Bug Gulch.....	Do.
Murie & Costello.....	No. 10.....	Bonanza.....	Do.
Stanley, L. V.....		do.....	Do.
Taylor & McLellen.....	No. 3.....	do.....	Do.
Whitman, Carl.....	No. 2.....	Little Eldorado.....	Do.
Wright & McNutt.....	No. 11.....	Bonanza.....	Do.

CIRCLE DISTRICT.

Adamik, Martin.....			Woodchopper.
Anderson, J. P.....			Miller House.
Armstrong, Charles.....			Woodchopper.
Bayless, Otto G.....			Deadwood.
Beatson & Nelson.....			Woodchopper.
Berry, C. J.....			Miller House.
Bloom, Pete.....			Deadwood.
Boyle, Charles.....			Woodchopper.
Boyle, John.....			Do.
Clark, A. P.....			Miller House.
Cochran, John Doe.....			Woodchopper.
De Michael, Michael.....			Circle.
Erickson, Riley.....			Deadwood.
Fenlanson, Chris.....			Woodchopper.
Fisher, August.....			Do.
Garner & Berry.....			Miller House.
Gibbon, Billy.....			Woodchopper.
Greep, Harry.....			Deadwood.
Holstrum, John.....			Woodchopper.
Ingalsbe, Al.....			Deadwood.
Kelly, J. F.....			Miller House.
Kronjaeger, Alfred.....			Deadwood.
Larson, Chris.....			Woodchopper.
Lee, O. B.....			Do.
Marigold Mining Co.....			Miller House.
Matthews, Dave.....			Woodchopper.
Morel & Johnson.....			Deadwood.
Phillips, Thos.....			Do.
Pompel, Jos.....			Do.
Powers, George.....			Woodchopper.
Sather, Antone A.....			Deadwood.
Soesniak, Frank.....			Woodchopper.
Slaven, Frank.....			Do.
Van Bibber, Pat.....			Deadwood.

COPPER RIVER DISTRICT.

Bergman, Fred.....		Big Four Gulch.....	Dempsey.
Brooks, E. W.....		Jolly Gulch.....	Nasina.
Canalle Bros.....		Big Four Gulch.....	Dempsey.
Carvey, Bert.....		Rex.....	Nasina.
Dan Creek Mining Co.....		Dan.....	McCarthy.
Elmer, J. M.....		Slate.....	Dempsey.
Estorly & Andrus.....		Chititu, Rex.....	Nasina.
Grosch, John.....		Miller Gulch.....	Dempsey.
Holmes & Brail.....		Upper Chesna.....	Do.
Kraemer, Hemple & Leavell.....		Slate, Miller Gulch.....	Do.
Miller, John S.....		do.....	Do.
Meyer & Prolig.....		Slate.....	Do.
Schroeder, William.....		Miller Gulch.....	Do.

CROW CREEK DISTRICT.

Alaska Crow Creek Mining Co.....		Crow.....	Girdwood via Anchorage.
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TABLE 10.—Data on placer mines in Alaska, 1916—Continued.

FAIRBANKS DISTRICT.

Operator.	Claim.	Creek.	Local address.
Atwood, Harry	No. 11	Goldstream	Fox.
Bean, Z. C.	No. 3	Wolf	Cleary.
Bleeker, Fred	No. 4	Goldstream	Fox.
Bostrom, Gus	No. 5	Cleary	Cleary.
Buren, Isaac	No. 6	do.	Do.
Burns & Gorgens		Head of Chatam.	Do.
Canning, James	No. 3, 2d tier.	Dome	Oines.
Casalegno Brothers			Fairbanks.
Christensen, Sam	Gold Hill	Ester	Berry.
Connelly, Frank	No. 15 below	Cleary	Chatanika.
Cook, Henry	Oregon Assoc.	Vault	Vault.
Cook, Nick	No. 11 bench	Goldstream	Fox.
Dixon, William	No. 15 below	Cleary	Chatanika.
Driscoll Brothers	No. 14 below	do.	Do.
Edmonds & Broadhurst	No. 1 below	Little Eldorado	Eldorado.
Erickson & Anderson	No. 3 below	do.	Do.
Erickson, Charles	No. 2 above	Wolf	Cleary.
Foster, W. H.	Fraction between No. 11 and No. 12 below.	Cleary	Chatanika.
Fraker & Nelson	No. 7 above	Little Eldorado	Eldorado.
Froese, A. C.	No. 7 below	Cleary	Cleary.
Fuller & Dunlap	Crackerjack bench	Chatanika River	Chatanika.
Griffin & Grille	No. 5 below	Fairbanks	Meehan.
Guis, Charles		Gilmore	Gilmore.
Guis, Louis	No. 5	Pedro	Do.
Hanot, Charles	No. 1 below	do.	Golden.
Hanson & Nelson	Serpentine Assoc.	Chatanika River	Chatanika.
Hess, Hans	Eldorado Assoc., Middle.	Little Eldorado	Eldorado.
Hilty, Al	No. 1 below	Cleary	Cleary.
Holm, Gust	Gold Hill	Ester	Berry.
Holmlund, A. J.	No. 2, tier bench	Chatanika River	Chatanika.
Hoover & Larson	Hope Bench	do.	Do.
Housdale & McDonald	No. 3 above	Little Eldorado	Eldorado.
Iverson, O.	No. 5 below	do.	Do.
Johnson, George	No. 13 below, 2d tier bench.	Cleary	Chatanika.
Johnson, Ludvig	No. 17	Goldstream	Fox.
Kallen & Dalton	Discovery	Little Eldorado	Eldorado.
Keys & Sons, E. M.	No. 1	Ready Bullion	Berry.
King & Nelson		Pedro	Gilmore.
KleinSmith, Victor (see Niggerhead Assoc.)			
Leach, John	No. 13 below	Cleary	Chatanika.
Leach & Bernard	No. 1 above	Ester	Berry.
Leedy, William	No. 4 above	do.	Do.
Letender, John	No. 3 above, 1st tier	Dome	Oines.
Lokka, John	No. 3 below	do.	Do.
Malsingeth, M.	Nigger Baby Fraction	Cleary	Chatanika.
Martin, M. J.	No. 6 below	Fairbanks	Meehan.
Martin & Blake	No. 15 below, 2d tier bench.	Cleary	Chatanika.
Marx, Guy	No. 9	Goldstream	Fox.
Morris, T.	No. 17 below	Cleary	Chatanika.
McPike, John	Discovery	Goldstream	Gilmore.
Nelson, Frank	No. 1 above	Little Eldorado	Eldorado.
Niggerhead Association		Dome	Oines.
Nerland, C. O.	No. 2 above	do.	Do.
Oberg & Furland	No. 3 below	Dome Creek Bench	Do.
Parker, Fred	No. 3 above	Fairbanks	Meehan.
Pearson & Jacobson	No. 5 below	do.	Do.
Pearson & Johnson	No. 6 below	Cleary	Cleary.
Quirk, Tom	No. 11 below	Dome	Oines.
Raymond & Carey	No. 11 below	Cleary	Chatanika.
Rehn, Theodore		Little Eldorado	Eldorado.
Rogge, Leo W.	No. 6 above	Cleary	Cleary.
Sagen, Alois	Paystreak Fraction	Ester	Berry.
Sather, Martin	No. 12 below	Fairbanks	Meehan.
Scannell, Tim	No. 14 below, 1st tier bench.	Cleary	Chatanika.
Shon, C. E.	No. 12 below	do.	Do.
Sjolaeth, J. L.	Top of the Ridge	Ester	Berry.
Skottland, Tom	No. 5 below	Little Eldorado	Eldorado.
Smith & McGlone	Mohawk Extension	St. Patricks	Berry.
Soderbloom, Gus	No. 15 below	Fairbanks	Meehan.
Stewart, Tool, Powell & Williams	No. 18 below	do.	Do.

TABLE 10.—Data on placer mines in Alaska, 1916—Continued.

FAIRBANKS DISTRICT—Continued.

Operator.	Claim.	Creek.	Local address.
Stutt, George.....	Gold Hill.....	Ester.....	Berry.
Sullivan & Olson.....	No. 1 above.....	do.....	Do.
Sundine, John.....	No. 6 below.....	Fairbanks.....	Meehan.
Takke, Isaac.....	No. 4 below.....	Dome.....	Oines.
Thompson, A. D. ^a	Steer Bench.....	Chatanika River.....	Chatanika.
Thomas, Theodore.....	No. 12.....	Goldstream.....	Fox.
Wagner, Henry.....	No. 6.....	do.....	Do.
Weiss, Sam.....	No. 14 below, 3d tier bench.....	Cleary.....	Chatanika.

HOT SPRINGS (MANLEY) DISTRICT.

Boor, A.....		Deep.....	Tofty.
Brandstrom & Anderson.....		Eureka.....	Eureka.
Frank & Graham.....		Pioneer.....	Do.
Glenn Mining Co.....			Do.
Howell & Cleveland.....		Woodchopper.....	Tofty.
Hosler, D. J.....		American.....	Hot Springs.
Johnson, Ed.....		Eureka.....	Eureka.
Jones & Stewart.....		Cash.....	Tofty.
Kanally & Hastings.....		American.....	Hot Springs.
Lane, A. H.....		Eureka.....	Eureka.
Murray, Michael.....		American.....	Hot Springs.
McKinzie, J.....	Gold Age.....	Miller.....	Tofty.
Ness, Edward.....		American.....	Hot Springs.
Olsen & Everson.....		Eureka.....	Eureka.
Peterson & Davidson.....		Boulder.....	Tofty.
Stevens, Frank.....		Eureka.....	Eureka.

IDITAROD DISTRICT.

Beatson, Bates, Longtin & Dawson.....	No. 1 below.....	Otter.....	Flat City.
Brevis, J. L.....		Willow.....	Do.
McKinzie & Mathewson.....		Chicken.....	Do.
McMillan, R.....	Discovery Bench.....	Otter.....	Do.
Manley, Frank.....		Willow.....	Do.
Rilley & Marston.....		Otter & Black.....	Do.
Strandberg, Dave.....	Link, Madden Bench, and Up-grade Placer.....		Do.
Welch, Al.....		Happy.....	Do.

INNOKO DISTRICT.

Greer & McNulty.....		Boob.....	Cooper.
Harling, Tom.....		Yankee.....	Ophir.
Pitcher & Van Orsdale.....		Tolstol.....	Cooper.
Reich, John.....	No. 5 below.....	Gaines.....	Ophir.
Schwasball, Andy.....		Tolstol.....	Cooper.
Snalley Brothers.....		Ophir.....	Ophir.
Spencer, Fred.....		Gaines.....	Do.
Thorn & Higgins.....	Hippard Fraction.....	do.....	Do.
Vibb, Nels.....	No. 2 above.....	do.....	Do.
Warren & Coutts.....	No. 11 above.....	do.....	Do.

KENAI PENINSULA.

Alaska Securities Corp.....			Seward.
Mathison Mining Co.....			Hope.
Pearson, A. & H.....			Do.
Renner, John.....			Sunrise.
St. Louis Mining & Trading Co.....			Hope.

^a Not mining.

TABLE 10.—Data on placer mines in Alaska, 1916—Continued.

KOYUKUK RIVER.

Operator.	Claim.	Creek.	Local address.
Collins, Ernest M.			Wiseman.
Holzer & Wilson Mining Co.			Do.
Pingel, H.			Do.
Smith, Ellingson & Nelson			Do.
Watts, Vernon			Do.
Webster & Co., Daniel			Do.
Williams, Mrs. Mary			Do.
Wool, John E.			Do.

MARSHALL (WADE HAMPTON) DISTRICT.

Anderson, Johnsen & Dahl	No. 1.	Fox Gulch	Fortuna Ledge.
Betsch, Jean, McGrath & McDonald	No. 1 above	Willow	Do.
Hockman & Webber	Lower Discovery	do.	Do.
Mack & McKinsie	Upper Discovery	do.	Do.
Nelson, Nels.	No. 4 above	do.	Do.
Pilcher, George M.		Elephant	Do.
Smith & Glacierio	No. 2 above	Willow	Do.
Waskey, Frank	Bumblebee	do.	Do.

RUBY DISTRICT.

Anderson & Johnson	No. 3 above	Long	Long City.
Bittle & Lindegard		Spruce	Poorman.
Black & Leveridge	No. 4	Flat	Do.
Buckley Brothers	Novikaket Assoc.	Long	Long City.
Cook, Charles		Poorman	Poorman.
Coyle Bros. & O'Donnell		do.	Do.
Coyle, John		Spruce	Do.
Felton, Alex.	No. 3	Tenderfoot	Do.
Gidling & Anderson	No. 2	Bear Pup	Long City.
Graham & Walker	Novikaket Assoc.	Long	Do.
Hegstrom & Nelson	No. 1	Straight	Do.
Herman & McKinnon	No. 2	Duncan	Poorman.
Jones & Lundin	Alabama Assoc.	Birch	Long City.
Kells, A.	Windy Bench	Long	Do.
Kigbush, Davis & Olson	Hagan Fraction	Poorman	Poorman.
LaBelle, Joseph		Tamarack	Do.
Larson, Alex.	No. 1 above	Long	Long City.
Matheson, Griund & Wick		Spruce	Poorman.
Miller & Pike	Wedge Fraction between No. 2 and No. 3.	Bear Pup	Long City.
Monahan, John		Spruce	Poorman.
McCloud, D.	Windy Claim	Flat	Do.
Nixon & Manual		Poorman	Do.
Nelhoff, Knute & Ditz		Spruce	Spruce.
Salch, Walker & Ditz	Mascott Bench	Long	Long City.
Shorpshear, J. F.	No. 2 Bench	Poorman	Poorman.
Strite, Charles	No. 3	Bear Pup	Long City.
Swanson & Cale		Fourth of July	Do.
Thompson, Morton, Johnson & McLaoughlin	Banner	Flat	Poorman.
Van Winkle & Wallace	Buckeye Bench	Long	Long City.
Ward, Drake & Kells	Novikaket Assoc.	do.	Do.
Ward & Bishop		Tamarack	Poorman.
Willeke, Herman	Surprise Fraction	Poorman	Do.
Wyman & Balanger	O. K.	Head of Flat	Do.
Yukon Gold Co.		Greenstone	Long City.

SEWARD PENINSULA.

Arctic Mining Co.		Bangor	Nome.
Brown, Robert		Sweepstake	Haycock.
Candle Ditch Co.	Bully Hill		Candle.
Carlson, Gabe		Tundra near Little	Nome.
Clark, Harry		Dexter	Do.
Coggins, Bernard		do.	Do.
Connely & Jensen		Gold Bottom	Do.
Connely & Bros., Tom		Dexter	Do.
Cordova, A. V.	Bessie Bench	Holyoke	Do.
Dakota-Alaska		Boulder	Do.
Fairhaven Mining Co.		Innamchuk River	Deering.
French, A. E.		Jump	Candle.

TABLE 10.—Data on placer mines in Alaska, 1916—Continued.

SEWARD PENINSULA—Continued.

Operator.	Claim.	Creek.	Local address.
Gewiler & Gloor.....		Dime	Haycock.
Gillette & McMillan.....		Center	Nome.
Gum, Henry.....		do	Do.
Gunderson, Lars.....		Dime	Haycock.
Hanson, Olson & Evans.....		Candle	Candle.
Jepson, Carl.....		Sweepstake	Haycock.
Kronquist, F. S.....		Candle	Candle.
Landstrom, A. J.....		Little	Nome.
Los Brothers & Halloran.....		Candle	Candle.
Lovell, C. O.....		Dime	Haycock.
Lundberg, O. A.....		Candle	Candle.
McCoy, F. J.....		Sweepstake	Haycock.
McGann, Thomas.....		Jess	Nome.
Madsen, Jens.....		Dime	Haycock.
Meles, Fred.....		Submarine Beach	Nome.
Nelson, Nels.....	Diamond L.	do	Do.
Nordlund Brothers.....		Candle	Candle.
Olson Brothers.....		Dime	Haycock.
Olson, Otto W.....	Tundra Assoc.	Little	Nome.
Orlem & Brown.....		Dexter	Do.
Peterson, Peter X.....		Buster	Do.
Pioneer Mining Co.....		Little	Do.
Porter, Wallace.....		Dime	Haycock.
Ravenkilda & Jensen.....		Jess	Nome.
Reed, C. T.....		Mountain	Do.
Regling & Olson.....		Dime	Haycock.
Roberts, E. A.....		Crooked	Council.
Rouse, Thomas.....		Innachuok River	Doeing.
Ryden, A.....		Bear	Candle.
Smith, Samuel.....		Dime	Haycock.
Stewart, A. C.....		Sledge	Nome.
Stonehouse, John.....		Buster	Do.
Sunset Mining Co.....		Sunset	Teller.
Sutton, W. L.....		Dime	Haycock.
Swanberg & Lee.....		Osborne	Nome.
Valentine & Anderson.....		Dime	Haycock.
Vogel, Charles.....	Diamond L.	do	Nome.
Waring, Frank.....	do.	do	Do.
Watson, Thomas.....		Center	Do.
Wilson, Al.....		Sweepstake	Haycock.

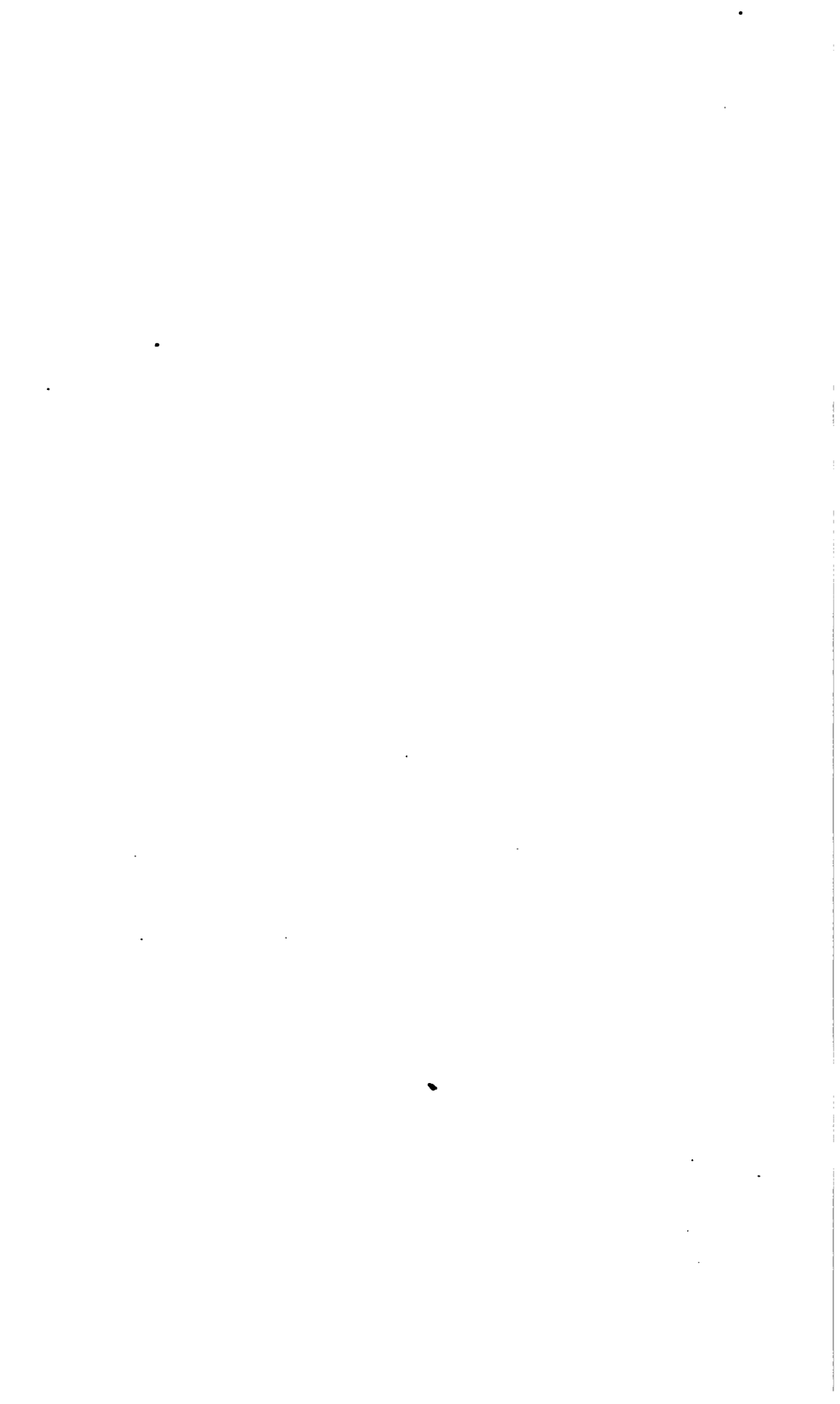
SUSITNA RIVER.

Adams, A. A.....		Falls.	Cache Creek via
Bubb & Babern.....		Dollar.	Susitna.
Carlson, C. J.....		Lucky Gulch.	Do.
Duff, Clark.....		Valdes.	Valdes Creek via
Eberhart & Anderson.....		Falls.	Gulkana.
Francis & Foster.....		Long.	Cache Creek via
Funk, J. C.....		Falls.	Susitna.
Gage & Mack.....		Thunder.	Do.
Gledakan, Wm.....		Bird.	Do.
Gray, John.....		Treasure.	Do.
Hammersmith, Chris.....		Bird.	Do.
Hansen, Chris.....		Willow.	Do.
Harper Bros.....		Nugget.	Do.
Jenkins, Frank.....		Willow.	Do.
Kast, Nelson & Larsen.....		Poorman.	Do.
McElroy & Rammer.....		Falls.	Do.
Peterson, William.....		Cache.	Do.
Price, Hugh.....		Nugget.	Do.
Raymond, Carl.....		do.	Do.
Rice, John.....		Willow.	Do.
Richardson, R.....		Ramsdyke.	Do.
Smith & Hogan.....		Nugget.	Do.
Tesmer & Bledermann.....		Cache.	Do.
Thunder Creek Mining Co. (M. A. Ellis, manager).		Thunder, Windy..	Do.
Valdez Placer Mines Co.....		Valdes.	Valdes Creek via
Van Eldenstein & Zindel.....		Peters.	Gulkana.
Weatherall & Andersen.....		Gold.	Cache Creek via
Wolf & Maloche.....		Spruce.	Susitna.
			Do.

TABLE 10.—*Data on placer mines in Alaska, 1916—Continued.*

TOLOVANA DISTRICT.

Operator.	Claim.	Creek.	Local address.
Allendale, William.....	Hidden Treasure	Livengood.....	Brooks.
Craig & McKinnon.....	Association.	do.....	Do.
Crook, Henry.....	Ready Bullion.....	do.....	Do.
Dooling & Fater.....	Discovery Bench.....	do.....	Do.
Johnson & Bostrom.....	Gold Dollar Association.	do.....	Do.
Jorgenson & Patton.....	Eldorado Bench.....	do.....	Do.
Kelly, Sampson & Colbert.....	Gold Dollar Association.	do.....	Do.
Kattleson & Curry.....	No. 17 above.....	do.....	Do.
Koebech & Eagler.....	No. 16 above.....	do.....	Do.
Lind, Pierce & Cramer.....	Gan Association.....	do.....	Do.
Paterson & Finley.....	Deep Channel Association.	do.....	Do.
Peterson & Anderson.....	Marriettes Bench.....	do.....	Do.
Peterson & McKinnan.....	Gold Dollar Association.	do.....	Do.
Peterson & Nelson.....	Virginia Association.	do.....	Do.
Price & Carnos.....	Eagle Bench.....	do.....	Do.
Revis, Nash, Miller & Clausen.....	Fish Association.....	do.....	Do.
Shaw & Doggett.....	Sunnyside Bench.....	do.....	Do.
Zuber & Altman.....	do.....	do.....	Do.



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DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

**MINING AND MILLING OF LEAD AND ZINC ORES IN
THE MISSOURI-KANSAS-OKLAHOMA
ZINC DISTRICT**

BY

CLARENCE A. WRIGHT

COOPERATION WITH THE MISSOURI BUREAU OF MINES AND GEOLOGY

H. A. BUEHLER, State Geologist



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MINING AND MILLING OF LEAD AND ZINC ORES IN THE MISSOURI-KANSAS-OKLAHOMA ZINC DISTRICT.

By CLARENCE A. WRIGHT.

INTRODUCTION.

The Missouri-Kansas-Oklahoma lead and zinc district, better known as the Joplin district, includes the mines in southwest Missouri and in those parts of Kansas and Oklahoma that are directly adjacent. (See Pl. I.) This report of investigations carried on by the Bureau of Mines gives the methods used in mining and milling and indicates in some detail the conditions that affect the efficiency of those methods; it does not attempt to discuss the geology of the district, except incidentally, as this has been described in numerous reports by several geologists. Some of the more important recent publications are those of the United States Geological Survey by Smith and Siebenthal,^a Bain,^b and also those of the Missouri State geological survey by Buckley and Buehler.^c Other reports are those by Schmidt and Léonard,^d and Winslow,^e of the Missouri geological survey, by Haworth and Crane,^f of the Kansas geological survey, and by Jenney.^g

Although the lead and zinc ores are closely associated, the proportion of zinc ore mined at present is far greater than in former years. For this reason and because the losses in milling zinc ores are proportionately greater than in milling those of lead, the zinc ores are here given most consideration. Only the mines and mills actually examined and tested are discussed in detail, statements of

^a Smith, W. S. T., and Siebenthal, C. E., Joplin district folio (No. 148), Geol. Atlas, U. S. Geol. Survey, 1907, 20 pp. Siebenthal, C. E., Origin of the zinc and lead deposits of the Joplin region, Missouri, Kansas, and Oklahoma: U. S. Geol. Survey Bull. 606, 1915, 283 pp. Smith, W. S. T., Lead and zinc deposits of the Joplin district, Mo.-Kan.: U. S. Geol. Survey Bull. 213, 1903, pp. 197-204.

^b Bain, H. F., Preliminary report on the lead and zinc deposits of the Ozark region: Twenty-second Ann. Rept. U. S. Geol. Survey, pt. 2, 1901, pp. 111-215.

^c Buckley, E. R., and Buehler, H. A., Geology of the Granby area: Mo. Bureau of Geology and Mines, vol. 4, 2d ser., 1906, 120 pp.

^d Schmidt, A., and Leonard, A., Lead and zinc regions of southwestern Missouri: Mo. Geol. Survey, vol. 1, 1874, pp. 381-502.

^e Winslow, A., Lead and zinc deposits: Missouri Geol. Survey, vol. 6, 7, 1894.

^f Haworth, Erasmus, Crane, W. R., Rogers, A. F., and others. Special report on lead and zinc: Univ. Geol. Survey Kansas, vol. 7, 1904.

^g Jenney, W. P., Lead and zinc deposits of the Mississippi Valley: Trans. Am. Inst. Min. Eng., vol. 22, 1894, pp. 171-225, 642-646.

results being restricted to the mines in general, with no special reference to any one mine.

Several mill tests were made to determine in a general way the efficiency of the milling practice in this district and to discover at what stage of the process the losses were greatest. Making these mill tests proved difficult, for, because of the arrangement of the mills and the necessity of estimating the tonnage treated, systematic mill testing and sampling can be done at only a few plants. Many tables giving results of mill tests and screen tests have been omitted because they would serve only to emphasize the important points shown by other data given.

The writer has tried to bring out the possibilities of making certain improvements that would effect a greater saving, and has endeavored to avoid the use of technical terms and phrases in order that all statements may be clear and easily understood.

ACKNOWLEDGMENTS.

The writer is greatly indebted to H. A. Buehler, State geologist of Missouri, who not only offered many suggestions, but generously contributed the parts on "Geology" and "Prospecting;" to G. B. Corless, formerly of the Missouri bureau of mines and geology, who prepared the map of the district and assisted in many other ways; and to R. R. Rinker, formerly chemist of the Missouri bureau of mines and geology, for many chemical analyses, only a part of which are presented in this report.

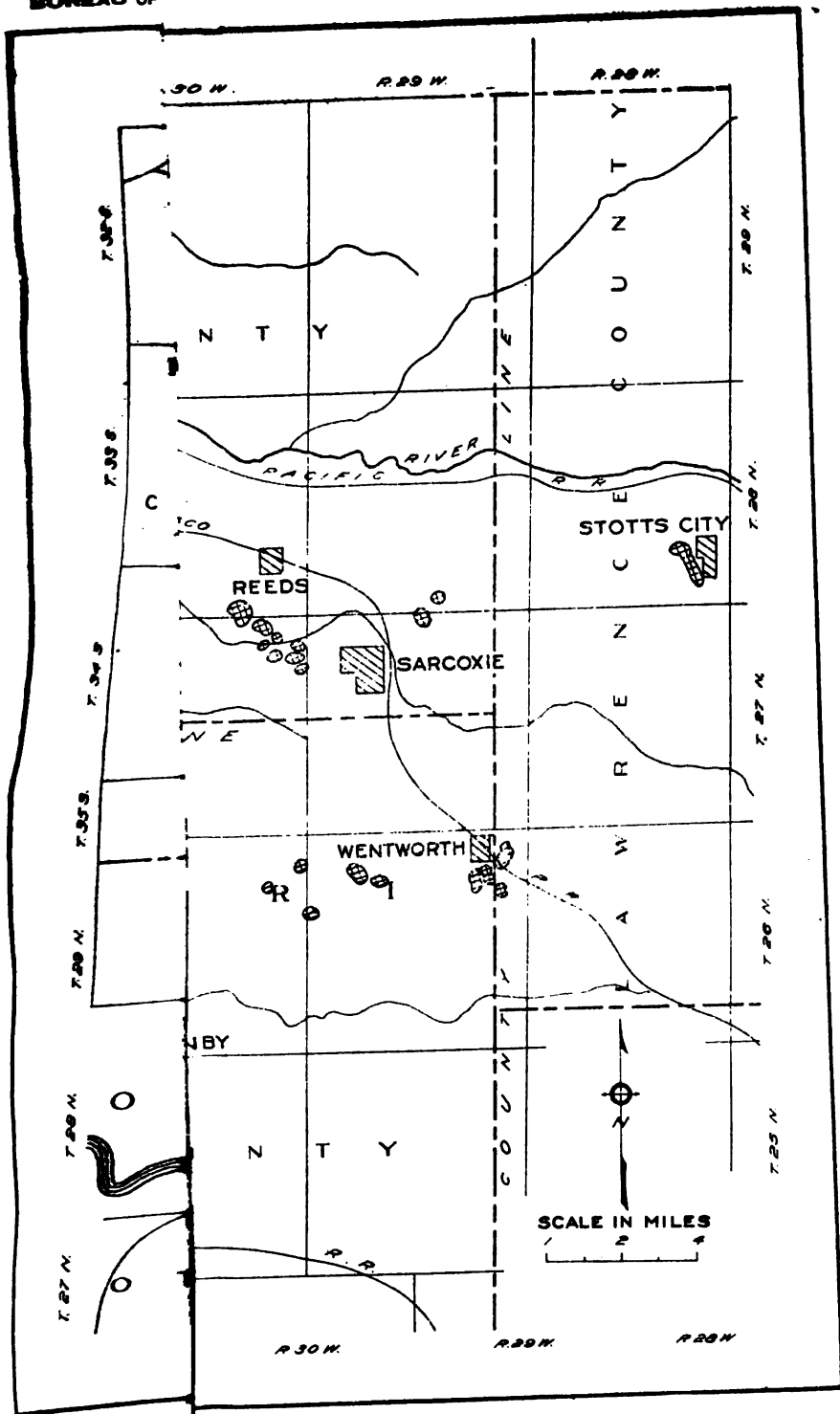
Acknowledgment is also due to operators and mine superintendents for furnishing information, for extending many courtesies and privileges, and for giving assistance whenever possible.

GEOLOGY OF ORE DEPOSITS.

In general the district has a gently rolling, prairie-like surface, although in the immediate vicinity of the larger streams the topography is more broken and frequently becomes decidedly rough.

The ore bodies of the district have definite relations to certain geologic formations and the following brief summary of the stratigraphy is intended simply as a key to the beds encountered in prospecting.

The formations underlying the Joplin district comprise about 1,800 feet of sedimentary beds, ranging in age from Cambrian to Pennsylvanian, resting upon igneous rocks of pre-Cambrian age. The generalized section shown in figure 1, is based upon the record of well No. 6 of the Carthage water works, Carthage, Mo.



DISTRICT.

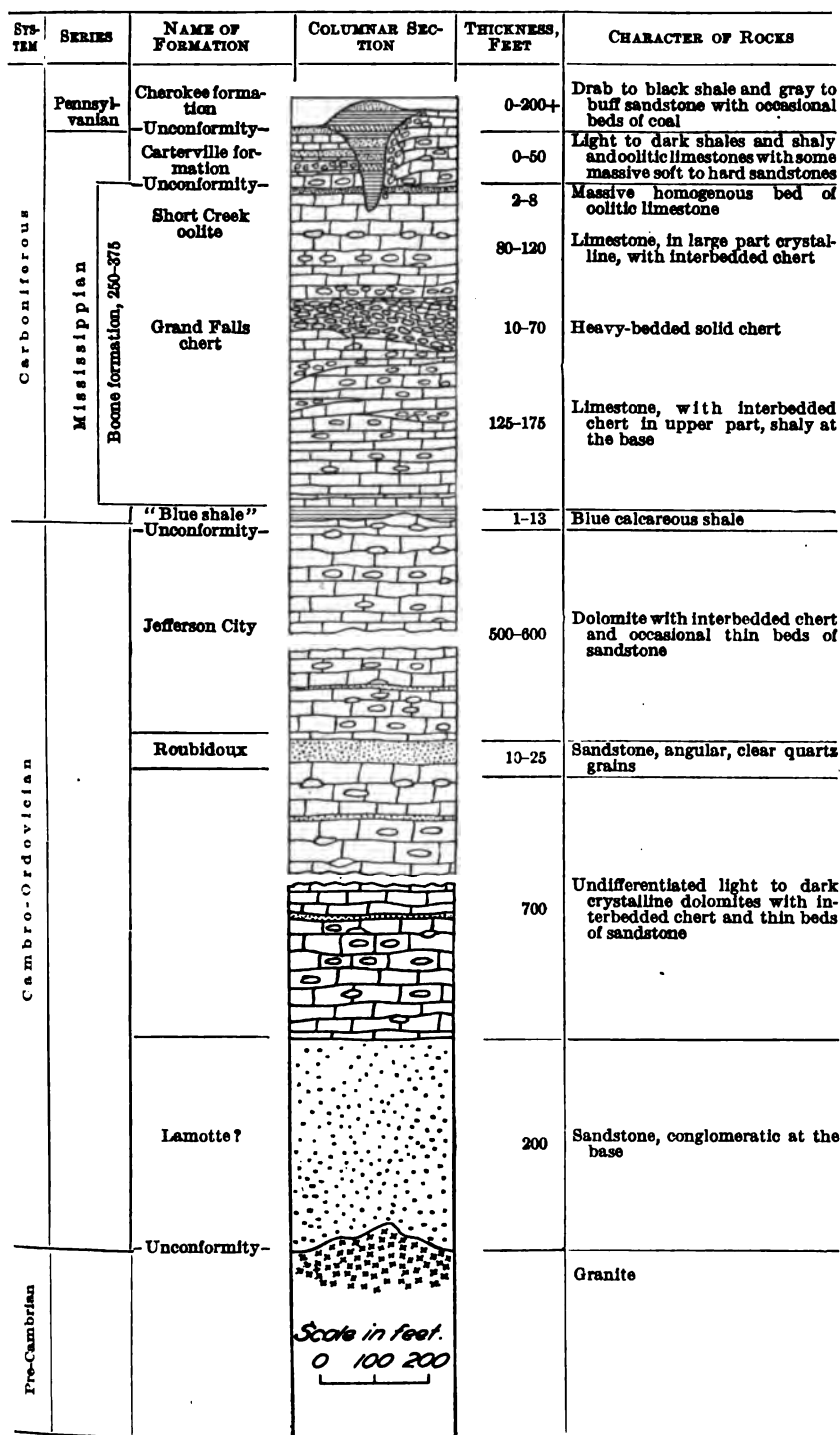


FIGURE 1.—Generalized section of formations in Joplin district.

DESCRIPTION OF BEDS.

The lowest stratified beds comprise 200 feet of sandstone which may be correlated with the Lamotte sandstone of the Ozark region. Overlying this are cherty and noncherty dolomites, approximately 700 feet thick, which have not been differentiated, although they correspond to the interval occupied by the Bonneterre, Davis, Derby, Doe Run, Eminence, Potosi, and Gasconade formations. The Roubidoux sandstone which overlies these beds, at 900 to 1,100 feet beneath the surface, is known to local drillers as the "sea level" or "water" sand and is the formation sought in the deep wells drilled for water supply. The Jefferson City dolomite lies above the Roubidoux and averages 540 feet in thickness.

Above the dolomite is a marked unconformity. In the southern part of the area a black shale (Chattanooga) of Devonian age overlies the dolomite, whereas in the northern part a blue shale of Mississippian age rests directly on it.

The Boone formation, of Mississippian age, consisting of cherty limestone and chert, overlies the shale and is the chief surface formation of the region; it is approximately 325 feet thick. Above the Boone is the Chester series, also of Mississippian age. On the western edge of the district where this series overlies the Boone almost continuously, it consists of limestone and sandstone; but to the east the Chester beds are chiefly in depressions or "sinks" formed during the erosion interval between the deposition of the Boone and the Chester.

The Cherokee shale of Pennsylvanian age is the youngest formation of the region; it is the surface rock along the western edge of the district, whereas to the east it forms outliers and occupies sinks or structural depressions similar to those occupied by the Chester, which it overlies.

ORE BODIES.

The ore bodies are chiefly in the Boone formation and their character varies with their position. The upper 100 to 150 feet of the formation is cherty, crystalline limestone, underneath which is 2 to 8 feet of oolitic limestone known as the Short Creek oolite. Beneath the Short Creek is 100 feet of cherty, coarsely crystalline limestone similar to that above. Underlying this division is 8 to 30 feet of chert known as the Grand Falls chert. The lower part of the Boone formation consists of 50 to 100 or more feet of bluish limestone and chert which rests upon the Mississippian or Devonian shale.

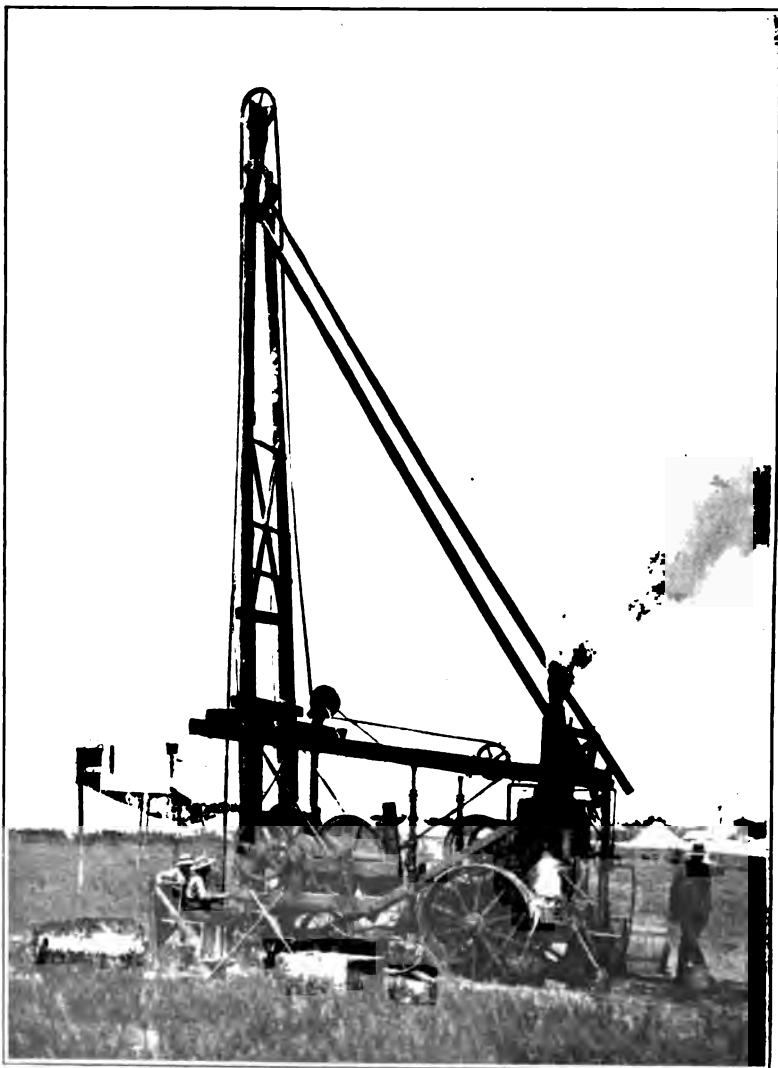
The ore bodies above the Grand Falls chert are usually irregular and lie mainly at the contact of the undecomposed Boone limestone and the shale or sandstone of the Cherokee, which occupies old depressions or solution channels in the limestone. The Grand Falls chert has been mineralized rather uniformly over large areas and



A. HAND WINDLASS USED AT PROSPECT SHAFT.



B. HORSE WHIM USED AT PROSPECT SHAFT.



PROSPECTING WITH CHURN DRILL.

hence is known as sheet ground. In parts of the area the limestone underneath the Grand Falls chert carries small mineralized fissures and openings and these deposits, where worked, have usually shown high faces of comparatively low-grade ore. In parts of the district the ore deposits extend into the underlying shale.

In the Miami, Okla., area the shallow deposits are in the Chester sandstones where these have been fractured so as to form a "bowl-dery" ground that in many respects is similar to that of some shallow deposits in the upper part of the Boone formation.

In general the Short Creek oolite and Grand Falls chert are the chief markers or key beds utilized in determining geologic horizons in prospecting. The Short Creek oolite is approximately 100 feet above the Grand Falls chert and in prospecting sheet ground may be used to indicate the depth of hole necessary. In searching for ore bodies above the Grand Falls it is usual to prospect the juncture of the Cherokee shale and Boone limestone as indicated by exposures of the two. In the Miami district the Cherokee formation can not be used as a guide, for it forms most of the surface.

PROSPECTING.

In the early development of the Joplin district, shallow deposits were sought by sinking shafts or test pits. (See Pl. II.) As the shallow ores were exhausted and mining extended below water level prospecting by shafts became too expensive and it has largely been superseded by drilling. (See Pl. III.)

Churn drills are used exclusively. The brecciated cherts and the many cavities prohibit the general use of the diamond drill. In the Oronogo Circle mine a shot drill was used underground to prospect the lower runs of ore. Near Springfield diamond drills have been used underground a little in prospecting the parallel "gumbo" runs in the shaly, noncherty formations of that area.

Much of the churn drilling is done by contract, the price ranging from \$0.90 to \$1.50 per foot. The maximum depth of hole is seldom more than 350 feet and the average is probably less than 200 feet. The usual diameter is 5½ inches. The holes are cased to solid rock. In soft ground the casing goes to the top of the ore body; if a lower run of ore is found a smaller casing is inserted to its top in order to prevent bits of ore from the upper run salting the drillings from the lower.

When ore is struck, samples of cuttings are taken every 2 feet. Usually the samples are collected by emptying the bailer into a bucket or half barrel and decanting the water after the drillings have settled. A part of the sample is placed in a pile on the ground near the drill, tags being used to show the depths of the beds from which the successive piles of cuttings came.

In some places the drillings are washed with water. This procedure, however, may materially affect the percentage of ore in the sample. In other places the drillings are carefully dried and sampled, although a small part of the various piles on the ground is often taken as a sample to represent the ground drilled through. Of course, such sampling is liable to spoil the value of the assays.

Commonly a driller designates the ore as "shines," "good shines," "fair jack," or "good jack." As the real significance of such designations depends upon the opinion of the driller, the future value of many records is virtually lost. Only by careful sampling and assaying of the cuttings can reliable records be had.

In order to verify a drill record a hole is sometimes shot. After the hole is thoroughly cleaned a small sheet-iron pipe containing dynamite is lowered to the ore and the charge exploded. The material loosened by the shot is then broken into particles with the drill and taken out with the sand bucket. This method has given some very reliable results.

Hand jigs are used to some extent in development work and at some small mines that are in the prospect stage. (Pl. IV.) This is especially true of shallow deposits and of prospectors with little working capital.

ORE DEPOSITS.

OCCURRENCE.

SHAPE OF ORE BODIES.

The lead and zinc ores are widely distributed throughout in rather well-defined blanket deposits, better known as "sheet-ground" deposits, or in irregular deposits known as "runs."

In the sheet-ground deposits the ore bodies lie nearly horizontal between the bedded layers of cherts. The seams of ore vary in thickness from a fraction of an inch to several inches, and here and there contain cavities lined with crystals of lead and zinc minerals, with minor amounts of iron sulphides and other associated minerals. The ore-bearing beds are 6 to 20 feet thick, and are more uniform in richness and less liable to terminate abruptly than the "runs." Brecciation is relatively unimportant in the sheet-ground deposits.

The "runs" and the other irregular bodies of ore may be divided into two classes—those in fairly hard ground, where little timbering is necessary, and those known as "soft-ground" deposits, which require timbering in nearly all the drifts. These ore bodies usually have an elongated form and many are of considerable length and thickness. A few deposits are circular or semicircular in shape and are known as "circle" deposits. The so-called runs and the irregular ore bodies are found at shallower depths than deposits of the sheet-ground type,



HAND JIGS. SUCH JIGS ARE STILL USED AT SOME SMALL MINES AND IN DEVELOPMENT WORK.

and are as a rule considerably richer in mineral content, often containing large pockets of nearly pure blende and galena. These minerals are commonly associated with a dark-colored secondary chert as the main filling material of fractures and cavities.

GANGUE MATERIALS.

Chert and dolomite are the most important gangue minerals. Limestone, shale, "soapstone," and mud are less important. In some of the deposits the chert and dolomite are intimately mixed, the zinc blende being embedded in a black secondary chert that cements the gangue and the blende. Chert, or flint, is the characteristic gangue material of the ore deposits and can be recognized in three different varieties: The white chert, the blue to gray chert, and the black or secondary chert. The blue chert is known as "live flint," and its occurrence usually indicates the presence of blende. The black variety, which has already been mentioned as filling fractures, cavities, and the interstices of the brecciated rock, is generally found in the brecciated areas closely associated with sphalerite, or in places with galena, and as a cementing material or matrix for the blue and white cherts with zinc blende. The so-called "cotton rock," a decomposed chert, dead white in color, often forms layers several inches thick near the surface or overlying the ore bodies. It is noticeable in the roof of some of the "sheet-ground" mines, where it usually has to be taken down as waste with the ore or has to be propped with posts at frequent intervals.

DISTRIBUTION OF OXIDIZED AND SULPHIDE ORES.

Generally the oxidized ores of zinc and lead lie above and near the level of groundwater, the sulphide of zinc becoming dominant below, the proportion of galena growing less, and that of iron sulphides greater as the depth increases. Zinc blende is seldom found in the weathered zone, for it becomes oxidized, generally to silicate or carbonate of zinc, by the action of surface waters. As the general course of these waters is downward, and as the blende is more soluble and more easily oxidized than galena, it is found at a greater depth, below the influence of the oxidizing agents. Hence, the finding of a large body of galena with blende scattered through it indicates the probability of a good deposit of blende being found below.

MINERAL CONSTITUENTS OF THE ORES.

Galena, cerussite, sphalerite, calamine, and smithsonite are the minerals of chief commercial importance. These minerals are locally better known as "lead," for galena; "blende" or "rosin jack," "ruby jack," "steel jack," or "blackjack" for sphalerite;

and "silicate" for calamine and smithsonite. Closely associated with these minerals are pyrite, marcasite, chalcopyrite, calcite, dolomite, and, in a few mines, small crystals of quartz. Local names are "mundic" for the iron sulphides, "tiff" for calcite, and "spar" for dolomite. Besides these, in a few of the mines there are small deposits of bituminous compounds, which are termed by the miners "coal tar" or "asphalt." These compounds, which are described in detail on page 12, and are discussed by Siebenthal,^a occupy small openings or crevices and are usually viscous at the ordinary temperature in the mines.

A yellow to light-brown clay is associated with the ore, especially in the "soft-ground" deposits; and this clay, although not a definite mineral, in many places contains considerable zinc silicate. Several other less common and less important minerals are associated with the ores.

DETAILED DESCRIPTION OF CHIEF MINERALS.

As the specific gravity and composition of the chief minerals have a direct bearing on the methods of concentration, these minerals are described separately. The physical and chemical properties of the minerals are taken from Dana.^b

SPHALERITE OR ZINC BLENDE.

Composition.—Pure zinc sulphide (ZnS)=33 per cent sulphur and 67 per cent zinc, but natural sphalerite often contains small quantities of iron, manganese, and cadmium, and rarely other impurities. The sphalerite of the Joplin district is unusually pure, containing little iron and cadmium.

Physical properties.—Specific gravity, 3.9 to 4.1; hardness, 3.5 to 4. Luster, resinous to adamantine. Color, commonly yellow to brown ("rosin jack"), and dark brown to black ("blackjack"); also red ("ruby jack"), and almost colorless when pure. Streak, white to yellowish-brown.

Sphalerite, or zinc blende, is the principal ore mineral of the Joplin district. It occurs as crystals and crystal aggregates lining cavities; and also massive in seams and in the chert breccia.

SMITHSONITE.

Composition.—Zinc carbonate (ZnCO_3)= carbon dioxide 35.2 per cent, zinc oxide 64.8 per cent. Content of metallic zinc, when pure, 52 per cent. The impurities, found in small amounts, are the carbonates of iron and manganese.

Physical properties.—Specific gravity, 4.30 to 4.45; hardness, 5. Luster, vitreous inclining to pearly. Color white, often grayish, greenish, brownish-white. Streak, white. Fracture uneven to imperfectly conchoidal. Brittle.

Smithsonite does not occur in large quantities, as compared with sphalerite, but is an important zinc ore in the district. It is called

^a See Siebenthal, C. E., Origin of the zinc and lead deposits of the Joplin region—Missouri, Kansas, and Oklahoma: U. S. Geol. Survey Bull. 606, 1915; pp. 205, 206.

^b Dana, E. S., Text book of mineralogy, 1908.

"silicate," as is calamine, and it is sold as such. The crystals and crystal aggregates are often found coating other minerals, especially sphalerite, or lining cavities. Both smithsonite and calamine are generally limited to the zone of weathering. It has been stated that these minerals are not usually found in paying quantity below a depth of 50 feet, and on account of their solubility in surface waters are seldom found at depths less than 15 or 20 feet.^a

CALAMINE OR ZINC SILICATE.

Composition.—Zinc silicate (H_2ZnSiO_3)=Silica, 25.0 per cent; zinc oxide, 67.5 per cent; water, 7.5. When pure, calamine contains 54.2 per cent metallic zinc.

Physical properties.—Specific gravity, 3.4 to 3.5. Hardness, 4.5 to 5. Luster, vitreous. Color, white; sometimes with a bluish or greenish shade; also, yellowish to brown. Transparent to translucent. Streak, white. Fracture uneven to subconchoidal. Brittle.

Calamine, commonly known as "silicate" in the district, is found in the zone of oxidation, usually associated with smithsonite. "Tallow clays" often contain more or less basic zinc silicate that has not crystallized. These clays are fine grained and rather soft, having a conchoidal fracture. They are yellowish brown to gray in color, and in the mines are soft and plastic.

GOSLARITE.

Composition.—Hydrous sulphate of zinc ($ZnSO_4 \cdot 7H_2O$).

Physical properties.^b—Specific gravity, 1.95 to 2.04. Hardness, 2 to 2.5. Luster, vitreous. Color, white, yellowish.

Goslarite is formed by the decomposition of sphalerite, but is a rather rare mineral on account of its solubility in water. It has been found in old drifts, usually where the sulphate of iron has been found, and is known as ferrogoslarite.

GALENA.

Composition.—Lead sulphide (PbS)=Sulphur, 13.4 per cent; lead, 86.6 per cent.

Physical properties.—Specific gravity, 7.4 to 7.6. Hardness, 2.5 to 2.75. Luster, metallic. Color and streak, pure lead-gray. Opaque. Fracture, flat subconchoidal or even. Very brittle.

Galena, locally known as "lead," is the most important lead ore found in the district, but is of less commercial importance than sphalerite. It occurs in crystals and crystal aggregates in cavities; also in massive form, in seams of chert, and in chert breccia.

The galena of this district is unusually pure and is nonargentiferous, containing practically no silver. The lead concentrates produced average nearly 80 per cent lead. Galena is found in varying proportions associated with sphalerite, and when in large quan-

^a Siebenthal, C. E., and Smith, W. S. T., Joplin district folio (No. 148): Geol. atlas, U. S. Geol. Survey, 1907, p. 12.

^b Brush, G. J., and Penfield, S. L., Manual of determinative mineralogy, 1906, p. 291.

ties it lowers the zinc recovery and the grade of the zinc concentrates produced from the mills. The galena, owing to its high specific gravity and mode of occurrence, is easily separated from other minerals by wet-milling methods.

CERRUSITE.

Composition.—Lead carbonate (PbCO_3)=Carbon dioxide, 16.5 per cent; lead oxide, 83.5 per cent; or lead, 77.5 per cent.

Physical properties.—Specific gravity, 6.46 to 6.57. Hardness, 3 to 3.5. Luster, adamantine, inclining to vitreous or pearly. Color, white, gray, grayish black. Transparent to subtranslucent. Streak, uncolored. Fracture, conchoidal. Very brittle.

Only small amounts of cerrusite are found at present in the mines of the district, and as a rule only near the surface. It occurs as crystal aggregates, but also in granular, massive, and compact forms.

PYRITE AND MARCASITE.

Pyrite.—Composition: Iron sulphide (FeS_2)=sulphur 53.4 per cent; iron=46.6 per cent. Physical properties: Specific gravity, 4.95 to 5.1; hardness, 6 to 6.5; luster, metallic; color, pale brass yellow; streak, greenish black to brownish black; opaque.

Marcasite.—Composition, same as pyrite. Physical properties: Specific gravity, 4.85 to 4.90; hardness, 6 to 6.5; luster, metallic; color, pale bronze-yellow, deepening on exposure; streak, grayish or brownish black; opaque; fracture uneven; brittle.

These two minerals, locally known as "mundic," are common throughout the ores of the district in small quantities. When present in more or less large amounts in the zinc ores, they are detrimental to the grade of concentrates produced. The difficulty in separating these sulphides of iron from blende is due to the fact that their specific gravities differ little from that of the sulphide of zinc.

The pyrite and the marcasite occur in crystals, crystal aggregates, and massive, and are found in rather large quantities closely associated with the lead and zinc sulphide ores, in the mines near Thom's Station and in the Miami, Okla., district. They are also found in small amounts in the country rock at much lower depths.

CHALCOPYRITE.

Composition.—Sulphide of copper and iron (Cu,FeS_2)=Sulphur, 35.0 per cent; copper, 34.5 per cent; iron, 30.5 per cent.

Physical properties.—Specific gravity, 4.1 to 4.3. Hardness, 3.5 to 4. Luster, metallic. Color, brass yellow; often tarnished or iridescent. Streak, greenish black. Opaque. Fracture, uneven. Brittle.

Chalcopyrite is present only in small amounts with the ores of the district and has not been found in commercial quantities. It occurs in small tetragonal sphenoids, and when these small crystals are on sphalerite, one finds parallel orientation of the crystal faces.

CALCITE.

Composition.—Calcium carbonate (CaCO_3)=Carbon dioxide, 44.0 per cent; lime, 56.0 per cent. Small quantities of magnesium, iron, manganese, zinc, and lead may replace the calcium.

Physical properties.—Specific gravity, 2.714 in pure crystals, varying somewhat widely in impure forms. Hardness, 3. Cleavage, highly perfect. Luster, vitreous to subvitreous, to earthy. Color, white or colorless; pale shades of gray and yellow; also brown and black when impure. Transparent to opaque. Streak, white or grayish. Fracture, conchoidal, obtained with difficulty.

Calcite, locally known as "tuff," is one of the most common minerals associated with the ores. It is the essential constituent of limestone, and occurs in crystals or crystal aggregates, and in granular form as a cement in chert breccia. In crystal form it is most frequently found lining cavities.

GYPSUM.

Composition.—Hydrous calcium sulphate ($\text{CaSO}_4 + 2\text{H}_2\text{O}$)=Sulphur trioxide, 46.6 per cent; lime, 32.5 per cent; water, 20.9 per cent.

Physical properties.—Specific gravity, 2.314 to 2.328 when in pure crystals. Hardness, 1.5 to 2. Luster, pearly to subvitreous; massive varieties often glistening, sometimes dull earthy. Color usually white; sometimes gray, or brownish yellow; impure varieties often black, brown, or reddish brown. Transparent to opaque. Streak white.

The gypsum is a product of the weathering of calcium carbonate, and is seldom found in the ores of the district, except in small quantities.

DOLOMITE.

Composition.—Carbonate of calcium and magnesium [$(\text{Ca}, \text{Mg})\text{CO}_3$]=Carbon dioxide, 47.9 per cent; lime, 30.4 per cent; magnesia, 21.7 per cent; or calcium carbonate, 54.35 per cent; magnesium carbonate, 45.65 per cent.

Physical properties.—Specific gravity, 2.8 to 2.9. Hardness, 3.4 to 4. Luster, vitreous, inclining to pearly in some varieties. Color, white, reddish, or greenish white, light gray or pale buff. Transparent to translucent. Fracture, subconchoidal. Brittle.

Dolomite occurs in granular massive form, and also in crystal aggregates lining cavities. As a rule, it is closely associated with or adjacent to the ore deposits. It is commonly known as "spar" or "pink spar" depending upon the shade of color. When this mineral is associated in considerable quantity with the ore, it is often difficult to make a high-grade zinc concentrate, as the specific gravity of dolomite is somewhat higher than that of flint, and small amounts will be found with the zinc concentrates.

BARITE.

Composition.—Barium sulphate (BaSO_4)=Sulphur trioxide, 34.3 per cent; baryta, 65.7 per cent.

Physical properties.—Specific gravity, 4.3 to 4.6. Hardness, 2.5 to 3.5. Luster, vitreous, inclining to resinous. Color, white; also inclining to yellow, gray, or

brown. Transparent to translucent, to opaque. Streak, white. Fracture uneven. Brittle.

Barium sulphate is not common in this district, and is found in small scattered quantities. It is more abundant, however, in the Central Missouri district, where it is found with sphalerite and is practically impossible to separate from the zinc ore by the usual wet methods, although it is believed that flotation could be used to good advantage. When it occurs with the ore in the Missouri-Kansas-Oklahoma region it is concentrated with the zinc and is found in the zinc concentrates.

QUARTZ (CHERT).

Composition.—Silica (SiO_2)=Oxygen, 53.3 per cent, silicon, 46.7 per cent.

Physical properties.—Specific gravity, 2.65 to 2.66. Hardness, 7. Luster, vitreous, splendent to nearly dull. Colorless when pure; often various shades of yellow, brown, green, gray, blue, black. Streak, white. Transparent to opaque. Fracture conchoidal to subconchoidal in crystallized forms, uneven to splintery in some massive kinds. Brittle to tough.

In this district quartz occurs massive in the cryptocrystalline form known as chert and in granular aggregates in jasperoid, also known as secondary chert. It also is found as minute crystals coating siliceous surfaces, either on chert or jasperoid. As chert it is one of the most common minerals of the district, and locally is best known as flint.

JASPEROID.

Jasperoid is a dark siliceous rock resembling chert and is often called secondary chert. When found it always accompanies the ores in the sheet-ground mines and is the matrix of the chert breccia in mines other than those in sheet-ground.

HYDROCARBON COMPOUNDS.

The hydrocarbon compounds sometimes found with the ores of this district are generally not definite minerals, but are mixtures, which by the action of solvents or by fractional distillation can be separated into two or more components. These hydrocarbons are mostly bituminous coal, and bitumen, and are found in small quantities in many of the "shale patches." When the bitumen, locally known as "tar," "coal tar," or "asphalt" by the miners, is viscous, it often interferes with the milling of the ore by sticking to the rolls. Bituminous material has been especially noted in the mines near Miami, Okla., although it has also been found in the mines near Chitwood, Mo., and in other parts of the Missouri-Kansas-Oklahoma region.

VALUATION OF MINE AND ORE DEPOSITS.

VALUATION OF MINES IN RELATION TO PROSPECTING.

From the uncertain extent and character of the ore bodies in the Joplin district and the lack of prospecting or development work, other than churn drilling, before mining starts, it would seem that the opening of a mine represents a hazardous investment. Fortunately, the sum of money needed to open a mine in this field is much less than in some mining districts where larger enterprises are undertaken, so that the loss from the failure of any one property is not great.

Many failures in mining enterprises in the Joplin field are due to overvaluation based on insufficient prospecting and development and to the payment of excessive royalties. There are other causes, but these will not be discussed here.

The value of a prospective mine in this district is usually based on the zinc and lead content of the churn-drill cuttings. Unless the drill cuttings are fairly rich in zinc, they are described by the drillers simply as "shines" or "fair zinc." When the ore seems to be in fairly payable quantities, the drill cuttings are assayed for zinc, and also for lead if the latter is present. Even with the assay results, however, it is hardly possible to ascertain the value of an ore body.

Several times shafts have been sunk and mills erected on the basis of only three or four drill holes that gave cuttings fairly rich in metal content. A few of these mines proved to be of value, whereas others were worthless.

Such instances cause some investors in the Joplin district, as well as in other districts, to regard mining as a purely speculative and hazardous enterprise, especially those who do not realize that mining enterprises should be based on business principles, and are not familiar with mining conditions.

As the value of a mine is determined by its ore reserves and the quantity of available ore blocked out, sufficient prospecting and development work previous to beginning mining and milling operations is not only important but essential. The amount of such work necessary will depend on the richness of the ore, the size and shape of the ore body, and its depth below the surface, the location of the property with regard to other mines in the vicinity, and the area controlled by the company. Another important detail that bears a close relation to development work is the size of the concentrating plant to be built for treating the ore.

The design and capacity of the concentrating plant will depend on the character and the richness of the ore, the size of the ore body, and the tonnage to be handled. In other words, the capacity of the mill is more or less dependent on the life of the mine, and on whether the profits would be greater by treating the ore with a larger plant in a

short time, or with a smaller plant during a longer period of time. The building of small and inexpensive plants in the Joplin district is largely a result of the ore bodies being generally small, irregular, and uncertain.

As the loss of time from lack of a continuous supply of ore is more costly with a large plant than with a small one, large central mills have not been considered feasible in this district. On the other hand, the writer believes that at many properties considerably more prospecting and development work would have shown the advisability of erecting a larger mill or one better equipped for saving more of the lead and zinc, especially in the treatment of the fine material.

In determining the value of a property many difficulties that the investor or shareholder does not always realize, or knows nothing about, are encountered. A few of these difficulties are the irregularity of the ore bodies, uncertainty in the estimates of the quantity of ore available, overvaluation of the ore body through insufficient prospecting and development work, and the changes in the market prices for the metals. In order to obviate these difficulties as far as possible, the investor or operating company must determine, as conditions permit, the following points: The extent of the ore deposit; the amount of prospecting and development necessary to delimit the ore; the quantity of ore available; the size and equipment of the mill, with reference to the character of the ore; the conditions to be met in actual practice on a commercial scale; and, at all times, safety in the capital invested.

PRESENT SYSTEM OF ROYALTIES AND EFFECT ON MINING AND MILLING PRACTICE.

Frequently a mine operator has two or more adjacent holdings leased from different landowners and has proved by development that the ore body is large enough to last several years. He is, however, not allowed to treat the ore mined on one lease in a mill on another lease, but is obliged to erect a mill on each property. It would therefore seem advisable to buy the land outright at a reasonable price, but as such purchase is not always possible, the operator is compelled to mine and mill the ore from each lease separately.

As the land owner seldom mines his own property but leases it for royalties of 10 to 15 per cent of the gross mineral output, and the property is again subleased one or more times, thus increasing the royalty to 20 or even 30 per cent, the operating company is compelled to use those milling methods by which the largest capacity possible from a given plant can be obtained. Consequently, in order to make any profit, milling efficiency is sacrificed and the mills are often greatly overcrowded.

Some of the operators and miners try to mine only the higher grade ore, leaving much of the lower grade ore that by itself would be unprofitable to mine. In other words, when the royalties are excessive, it is often the practice to gouge the ore. In consequence much lean ore is left underground where it represents a loss or total waste of mineral resources.

Some landowners and lessors do not seem to realize that the high royalties obtained through the subleasing practice are detrimental to their interests, as their financial returns are less in the end, and many of the gouged workings are left in such condition as to hinder seriously further mining and prospecting should the operating company be obliged to suspend work. Another result is that drifts are often left in such condition that the lives of miners are endangered. High royalties also tend to reduce wages, because lessees are compelled to work the mine as cheaply as possible, and consequently the labor is apt to be less efficient.

On the other hand, the present system of leasing has an advantage in that the land is constantly being prospected for rich ore bodies, and when a rich showing is struck on a lease, other persons seek to obtain a lease on adjacent land in order to find the continuation of the ore body.

In view of the above conditions, a system of royalties based on the net profits, or of reduced royalties under the present system, especially in subleasing, or a combination of the two, would seem feasible, and at many mines would result in more efficient mining and milling, thus improving the general conditions and placing the district on a firmer and more reliable basis.

PRODUCTION.

The Joplin district is one of the main ore-producing areas of the Ozark region, a dissected plateau occupying most of the southern half of Missouri and parts of Kansas, Oklahoma, and Arkansas; in this region and around its border are many important deposits of lead and zinc ore. Not only is the Joplin district the principal zinc-producing area of this region but its ores yield more spelter than is produced by any other district in the United States. A comparison of the total mine production of zinc in the United States for the years 1906 to 1916, by States, is given in Table 1 following.

TABLE 1.—*Mine production of zinc in the United States, 1907-1916, in short tons.^a*

State.	1907	1908	1909	1910	1911	1912	1913	1914	1915	1916
Arizona.....	114	339	2,989	2,742	2,281	4,379	4,714	4,896	9,110	9,836
Arkansas.....	474	605	510	994	664	748	478	608	3,209	6,815
California.....	116				1,404	2,173	529	195	6,547	7,628
Colorado.....	26,394	15,065	25,605	38,545	47,304	66,111	59,673	48,387	52,297	67,143
Idaho.....	3,493	19	676	2,802	4,170	6,953	11,587	21,006	35,077	43,253
Illinois.....	737	1,717	2,163	3,549	4,219	6,065	2,236	4,811	5,584	3,404
Iowa.....	145	395	35	96						21
Kansas.....	17,772	14,119	11,445	13,229	10,272	10,633	10,088	11,284	14,365	12,446
Kentucky.....			56	6	158	491	327	220	784	1,066
Missouri.....	116,752	107,404	130,162	128,589	122,515	136,551	124,963	105,994	136,300	155,960
Montana.....	122	820	4,680	15,819	21,905	13,459	44,337	55,790	93,573	114,630
Nevada.....	1,084	558	1,507	1,354	1,774	6,661	7,210	6,490	12,188	16,222
New Hampshire.....							15	6	24	8
New Jersey.....	69,369	67,165	82,510	68,678	77,445	69,755	72,156	74,253	116,618	110,696
New Mexico.....	375	1,788	6,543	9,044	5,119	6,783	8,262	9,202	12,702	18,262
New York.....					81				2,455	4,507
North Carolina.....						142	10			
Oklahoma.....	1,657	4,529	7,806	6,394	5,150	5,769	11,664	13,992	14,214	28,754
Tennessee.....	109	344	596	966	1,117	2,191	5,583	10,425	16,461	26,426
Texas.....	22	18				119	326	108	15	116
Utah.....	2,726	730	4,930	8,184	8,920	8,534	9,429	7,995	12,146	14,796
Virginia.....		705	58	794	1,032	249	2,719	174	1,267	2,276
Washington.....					10				122	547
Wisconsin.....	18,490	18,206	23,152	25,927	29,720	33,050	30,110	31,113	41,403	56,903
Total.....	259,951	234,526	305,423	327,712	345,260	378,816	406,416	406,959	586,491	701,966

^a Figures for 1907-1915 from Mineral Resources U. S. for 1915: U. S. Geol. Survey, 1916, pt. 2, p. 859. Figures for 1916 supplied by U. S. Geol. Survey.

The rank of the Joplin district as a zinc producer is shown by Table 2. This table shows the production of spelter from the zinc ores in Missouri and also the production for the Joplin district, which includes practically the total zinc production from Missouri, Kansas, and Oklahoma. The percentage of total production in the United States represented by Missouri alone and by the Joplin district is also given for the different years.

TABLE 2.—*Primary spelter production from zinc ores of the Missouri-Kansas-Oklahoma districts.*

Source.	1906	1907	1908	1909	1910	1911	1912	1913	1914	1915 ^a	1916 ^a
Kansas ^b , short tons.....	3,902	13,850	8,628	9,185	10,220	6,843	5,668	9,956	10,634	14,365	12,446
Missouri ^b , do.....	130,348	141,824	123,655	140,676	140,652	127,540	149,557	129,018	114,019	136,300	155,960
Oklahoma ^b , do.....		719	2,235	3,008	2,297	2,247	2,041	6,397	9,449	14,314	26,754
Total.....	134,250	156,393	134,518	152,869	153,169	136,630	157,266	145,371	134,102	164,979	197,162
Missouri, percentage of United States production.....	65.3	63.4	64.8	61.0	55.7	47.0	46.2	38.2	33.2	23.2	22.2
Joplin district, percentage of United States production.....	67.2	70.0	70.5	66.4	60.7	50.8	48.5	43.1	39.0	28.1	28.1

^a Figures are for mine production of zinc.

^b Figures for 1906-1915 from annual volumes of Mineral Resources of United States, U. S. Geol. Survey. Figures for 1916 supplied by U. S. Geol. Survey.

The production of spelter from the Joplin district ores has varied more or less during the past several years. The production from the Miami, (Okla.) camp has been on the increase since 1912, and it is believed that before long it will rival both the Joplin and Webb City camps in production.

The tenor of crude lead and zinc ore and concentrates produced in the Missouri-Kansas-Oklahoma zinc district for the years 1908 to 1916 is given in Tables 3 to 6.

TABLE 3.—*Tenor of crude lead and zinc ore and concentrates produced in southwestern Missouri, 1908-1916.*^a

	1908	1909	1910	1911	1912	1913	1914	1915	1916
SOFT GROUND.^b									
Total crude ore.....	2,584,990	3,006,789	3,275,705	3,217,166	3,856,045	3,745,400	3,179,830	4,004,900	4,711,700
Total concentrates in crude ore.....	5.2	4.6	4.4	4.5	4.21	4.56	4.61	3.82	3.38
Lead.....	5.2	4.6	4.4	4.5	4.21	4.56	4.61	3.82	3.38
Zinc.....	4.0	4.2	4.0	4.1	3.87	4.12	4.13	3.56	3.10
Metal content of crude ore.....	2.5	2.6	2.6	2.6	2.41	2.58	2.64	2.26	1.92
Lead.....	2.5	2.6	2.6	2.6	2.41	2.58	2.64	2.26	1.92
Zinc.....	4.0	4.2	4.0	4.1	3.87	4.12	4.13	3.56	3.10
Average lead content of galena concentrates.....	2.6	2.3	2.2	2.3	2.14	2.24	2.26	2.01	1.70
Average lead content of galena concentrates.....	79.3	79.7	79.2	79.9	79.5	78.6	78.6	79.2	78.0
Average lead content of lead carbonate concentrates.....	41.8	43.6	47.6	54.7	54.8	62.2	51.3	58.1	55.5
Average zinc content of sphalerite concentrates.....	58.1	59.3	59.1	58.9	58.3	57.5	57.9	57.7	58.1
Average zinc content of silicates and carbonates.....	42.0	38.5	35.9	40.4	33.8	38.8	38.6	39.3	39.4
Average value per ton:									
Galena concentrates.....	52.50	52.37	51.06	54.41	54.43	51.51	46.16	48.19	52.09
Lead carbonate concentrates.....	27.44	28.89	32.41	35.72	35.52	32.68	24.19	32.56	61.48
Sphalerite concentrates.....	34.48	40.85	39.40	40.80	49.99	39.97	38.65	72.68	77.94
Zinc silicates and carbonates.....	20.90	23.74	23.58	23.76	28.95	22.45	21.13	45.00	50.21
SHEET GROUND.									
Total crude ore.....	3,422,976	4,994,123	5,779,192	4,944,910	5,465,100	4,303,900	3,594,170	6,501,000	8,484,700
Total concentrates in crude ore.....	3.4	3.1	2.6	2.6	2.62	2.63	2.69	2.25	2.19
Lead.....	3.4	3.1	2.6	2.6	2.62	2.63	2.69	2.25	2.19
Zinc.....	2.8	2.6	2.2	2.5	2.16	2.16	2.23	1.91	1.86
Metal content of crude ore.....	2.2	2.0	1.6	1.64	1.62	1.63	1.65	1.38	1.35
Lead.....	2.2	2.0	1.6	1.64	1.62	1.63	1.65	1.38	1.35
Zinc.....	1.7	1.6	1.3	1.4	1.36	1.37	1.35	1.25	1.25
Average lead content of galena concentrates.....	80.0	8.03	79.9	79.9	79.6	79.0	79.3	76.4	76.5
Average zinc content of sphalerite concentrates.....	57.6	59.5	59.3	58.6	58.6	58.3	58.4	59.1	59.2
Average value per ton:									
Galena concentrates.....	53.70	54.41	52.01	56.50	55.83	51.43	47.38	54.63	55.62
Sphalerite concentrates.....	32.50	41.55	40.81	39.79	50.84	41.69	38.86	78.79	86.02

^a Figures for 1908 from Mineral Resources of United States, 1909, U. S. Geol. Survey, 1909, Pt. I, pp. 518-19; figures for 1909-1916 from Mineral Resources of United States for 1910, U. S. Geol. Survey, 1911, p. 641; figures for 1911-1915 from Mineral Resources of United States for 1915, U. S. Geol. Survey, 1916, p. 881; figures for 1916 from U. S. Geol. Survey Press Bull. 220, June, 1917, p. 7.

^b Includes also the small production of lead and zinc ores in central Missouri.

MINING AND MILLING OF LEAD AND ZINC ORES.

Total crude ore.....	short tons.....	677,722	596,786	704,046	761,465	744,254	590,300	540,500	709,400	774,400
Total concentrates in crude ore.....	per cent.....	4.7	4.2	4.3	3.2	3.2	4.0	4.37		
Lead.....	do.....	4.5	3.9	3.9	2.7	2.8	3.5	3.3		
Zinc.....	do.....	2.8	2.4	2.5	1.9	1.9	2.3	4.04	3.92	3.15
Metal content of crude ore.....	do.....	4	2	3	4	3	4	2.58		
Lead.....	do.....	4	2	2	4	2	4	26		
Zinc.....	do.....	2.4	2.2	2.2	1.5	1.5	1.9	2.33	17	20
Average lead content of galena concentrates.....	do.....	78.6	77.4	70.8	78.9	78.8	78.2	77.9	2.25	1.78
Average lead content of lead carbonate concentrates.....	do.....			62.3		64.5			74.9	74.3
Average zinc content of sphalerite concentrates.....	do.....	57.7	56.1	55.3	56.0	54.5	55.7	57.4	57.4	57.0
Average zinc content of silicates and carbonates.....	do.....	40.0	38.0	34.0	38.5	38.2	35.9	34.2	38.0	38.0
Average value per ton:	dollars.....		48.65	48.53	54.64	52.88	50.94	42.02	52.79	73.12
Galena concentrates.....	do.....	53.11	48.65	48.53	54.64	52.88	50.94	42.02	52.79	73.12
Lead carbonate concentrates.....	do.....	32.70	38.72	40.92	36.19	40.45	39.58	38.06	68.96	75.13
Sphalerite concentrates.....	do.....	11.17	16.38	13.70	23.90	25.30	16.13	16.40	35.52	39.20
Zinc silicates and carbonates.....	do.....									

^a Figures for 1908 from Mineral Resources, United States, 1909, U. S. Geol. Survey, 1909, Pt. I, p. 510; figures for 1909 and 1910 from Mineral Resources, United States, 1910, U. S. Geol. Survey, 1911, p. 622; figures for 1911 to 1915, from Mineral Resources, United States, 1915, U. S. Geol. Survey, 1915, p. 884; figures for 1916 from U. S. Geol. Survey Press Bull. 320, June, 1917, p. 4.

TABLE 5.—*Tenor of lead and zinc ore and concentrates produced in Oklahoma, 1908-1916.*^a

	1908	1909	1910	1911	1912	1913	1914	1915	1916
Total crude ore.....	225,164	317,574	228,212	224,450	293,500	561,100	708,920	891,800	1,400,900
<i>Atfani district.</i>									
Total concentrates in crude ore.....									
Lead.....	10.6	9.2	11.8	7.6	7.2	5.9	5.08		
Zinc.....	2.2	2.4	2.9	2.0	2.1	1.5	1.24	1.10	1.15
Metal content of crude ore.....	8.4	6.8	8.9	4.9	5.1	4.4	3.84	3.26	4.08
Lead.....	1.9	1.9	2.4	1.8	1.7	2.0	2.08		
Zinc.....	4.1	3.4	4.2	2.8	2.6	1.2	1.00	1.86	
Average lead content of galena concentrates.....	79.8	78.0	78.2	78.3	79.6	79.3	80.00	78.9	2.35
Average zinc content of spialerite concentrates.....	48.9	50.3	48.7	50.4	51.8	53.6	54.8	55.7	57.77
Average value per ton:									
Galena concentrates.....	53.28	51.92	51.76	53.55	54.42	52.91	47.19	54.60	53.94
Spialerite concentrates.....	21.56	31.21	29.32	26.50	36.80	31.80	32.66	60.48	74.80
<i>Quepaw district.</i>									
Total concentrates in crude ore.....									
Lead.....	2.8	3.7	4.0	3.4		2.1			
Zinc.....	2.5	2.2	3.7	3.0	1.9	1.3		1.12	
Metal content of crude ore.....	1.2	2.2	2.4	2.0		1.22	3.82	2.14	2.42
Lead.....	1.3	2.0	2.2	1.7		1.13			
Zinc.....	1.4	2.0	2.2	1.7	1.1	1.1	2.14	1.28	1.31
Average lead content of galena concentrates.....	78.9	78.3	78.2	79.3	79.5	78.0	78.0	78.8	77.0
Average zinc content of spialerite concentrates.....	58.8	57.9	57.9	57.0	57.8	58.8	58.2	57.5	54.2
Average value per ton:									
Galena concentrates.....	61.74	51.41	50.84	55.32	50.00	51.55	50.00	53.74	71.79
Spialerite concentrates.....	31.02	41.39	38.81	36.78	48.36	38.03	24.87	80.97	75.30

^a Figures for 1908 from Mineral Resources, United States, 1909, U. S. Geol. Survey, 1909, Pt. I, pp. 528-527; figures for 1909-10 from Mineral Resources, United States, 1910, U. S. Geol. Survey, 1911, p. 857; figures for 1911-1915 from Mineral Resources, 1915, U. S. Geol. Survey, 1916, p. 668; figures for 1916 from U. S. Geol. Survey Press Bull. 320, June, 1917, p. 8.

GENERAL NATURE OF MINING AND MILLING IN THE DISTRICT.

The methods used in mining and milling the ores in this district may seem crude and wasteful to one accustomed to the more elaborate methods employed in other mining districts, but careful study shows that they are well adapted to the existing conditions. The ore bodies vary widely in shape, richness, and extent, and it is difficult to calculate the probable yield of an ore body from drill-hole records, especially in other than blanket or sheet-ground deposits. Frequently, however, failure or loss on investment by the operating company has been due more to insufficient drilling and development work previous to erecting the concentrating plant than to variations of the ore body or to poor management.

The generally small size and irregular shape of the ore bodies have not encouraged the building of large, expensive plants. Economy in first cost rather than a higher saving by more elaborate and efficient equipment has been the prevailing consideration. Increased milling capacity at small cost is sought, a necessary course where the ore is so low in mineral content and the tonnage controlled by the operating company so small as at most mines in this district. Also the high royalties demanded have naturally led to small rather than large investments.

Although the controlling conditions mentioned persist with little change, both the mining and the milling practice have greatly improved during the past few years, and conditions at the mines are being bettered steadily. This result has largely come through the higher prices received for zinc and lead ores, making profitable the working of leaner deposits, and through a closer saving in mills by the use of more efficient methods. A few years ago ores with less than 4 per cent of recoverable mineral content were considered unworkable, but now many deposits of sheet-ground ore with a content of less than 3 per cent are being worked with a good margin of profit.

One of the most noticeable improvements throughout the district is the better saving of zinc and lead in the fine material. Where formerly only three or four concentrating tables were used in a mill, four or five times as many can now be found in later mills of the same capacity. The commercial saving possible by treating the finest material, or slimes, a product that for many years was wasted, is being realized more and more. The average recovery of zinc is about 65 per cent, and a few of the better equipped mills are making a recovery of about 70 per cent based on the zinc content of the ore and as high as 75 or 80 per cent when the lead recovery is included. This is good practice in view of the equipment of the mills and the tonnage that passes through them. However, there is still plenty of room for

improvement in many of the mills, and it is to be hoped that the percentage of saving will show an even more marked increase during the next few years.

MINING METHODS.

UNDERGROUND WORK.

The mining methods employed vary with the nature of the deposits and have been developed to suit the conditions in the various mines. As the ore deposits are relatively flat and lie at shallow depths, they are reached by vertical shafts sunk at frequent intervals. The depth of shafts usually varies from 100 to 300 feet, although the depth is sometimes less than 100 feet in soft-ground deposits and nearly 400 feet at some mines in the Miami district, Okla. In general, the ore bodies are worked from one level, as the ore-bearing deposits are

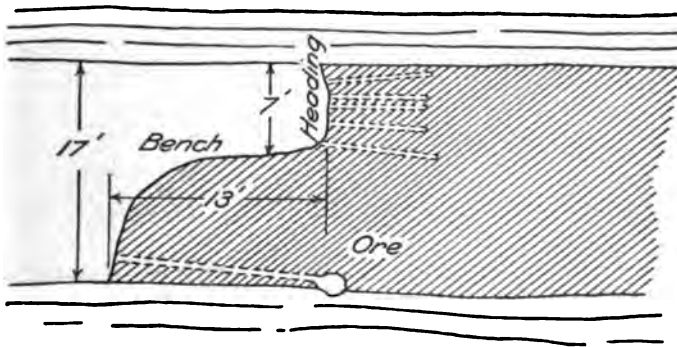


FIGURE 2.—Cross section showing underhand stoping as practiced in Joplin district.

rarely more than 30 feet thick. Where there are two distinct runs of ore, one above the other, separated by a relatively thick stratum of barren rock, the two runs are usually mined independently or separately, at different times (see Plate V).

Development work underground is usually limited, there being no marked distinction between it and the mining proper. The development work consists chiefly in sinking shafts to ore bodies that have been located by drill holes or by driving lateral drifts, and in some places raises or winzes are started in search of ore bodies indicated by mineral-bearing fissures or pockets. As a rule, the development work is not satisfactory, owing to the irregularity of the ore bodies and the lack of well-defined mineral-bearing zones.

Where possible, a system of underhand stoping (figs. 2 and 3) is practiced, especially in the "hard-ground" or "sheet-ground" mines, provided the working face is high enough to warrant stoping. Pillars are left at frequent intervals, the distance between centers and the thickness of the pillars depending upon the character of the

ore body and the height of face mined. The methods of mining used may be divided into "hard-ground" and "soft-ground" mining.

In hard-ground or "sheet-ground" mining, where the height of face warrants underhand stoping (see Pl. VI), a machine drill is usually set on a 7-foot column placed at the upper part of the working face, near the roof, in order to advance a heading about 6 to 8 feet high. When the heading or breast has been advanced 15 to 25 feet the underlying bench is drilled and blasted. This bench is usually 9 to 12 feet

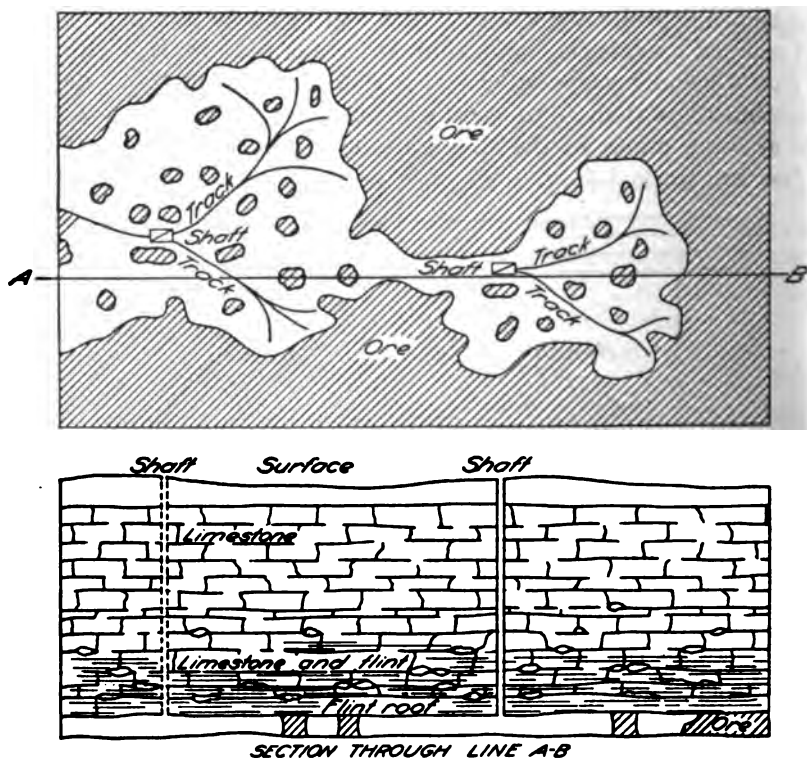


FIGURE 3.—Plan and section of "sheet-ground" mine in Joplin district.

thick, giving a total working face of 15 to 20 feet. If the thickness of ore should be more, or the face higher, than 20 feet, an intermediate bench is worked between the heading and underlying bench. In some of the hard-ground mines only the upper part of the ore-bearing deposit can be worked at a profit and in others only the lower part. This condition is chiefly due to the presence of an intermediate layer of practically barren rock, varying in thickness from 2 to 6 feet, which would have to be mined with the ore if the two ore-bearing strata were mined together. Where stoping is not carried on, or where the deposit is not more than 8 to 12 feet thick (Pl. VII), the



OLD WORKINGS. SHOWS UPPER AND LOWER WORKING LEVELS.



MINING BY BENCH SYSTEM.

and as to leave an irregular face to the best advantage drilling and blasting.

Where the deposits are "runs" or large and irregular the ore is often large enough to be worked from more than one level. In some mines this method has been used, but in many places the ore has been followed by raises from the bottom level, at which the mine was opened, to its upper limits, the broken ore being allowed to fall down a slope or chute whence it could be shoveled into cars. The method of mining used in such a case has been to follow the ore with a heading and break the underlying parts by the usual benching system. This method results in drifts or cuts 40 to 80 feet high and 20 to 40 feet wide.

Little timbering (see Pl. VIII, *A*) is done, except occasional props or posts placed wherever necessary to support slabs and to prevent possible scaling of the roof. However, pillars (see Pl. VIII, *B*) are left at intervals of 30 to 60 feet, center to center. They vary in thickness from 12 to 30 feet, depending on the nature of the deposit, the height of face, and the character and firmness of the roof and overlying strata. The average thickness of the pillars left to protect the shafts is about 50 feet. Wherever possible, the pillars are of the relatively leaner ore. The proportion of ground left as pillars is 15 to 30 per cent, although the pillars are usually trimmed or in some mines pulled after the mine has been worked out.

In the sheet-ground mines the usual roof is a layer of hard flint, 3 to 5 feet thick, which serves as a good support for the overlying beds and as a good bedding plane for the miner to work against. Where this layer is thin or lacking, more or less trimming of the roof is required after each round of shots. In a few of the mines a thin layer of white "cotton rock," or decomposed chert, 2 inches to 1 foot thick, is found in the roof. As this rock in places is relatively soft and can not be supported by posts, it has to be taken down and shoveled out with the ore, thus increasing the amount of "dead rock" to be handled and decreasing the relative richness of the ore treated.

"Soft-ground" deposits are mined by driving lateral drifts through the ore body, the walls and the roof being supported by timbering or by a combination of pillars and timbering. Usually the fore-poling system of timbering is used where the drifts are narrow and the face is relatively low, whereas square sets are used where the drifts are fairly wide and the face is relatively high. Lagging is used to prevent small pieces of rock falling into the drifts.

SHAFTS.

As the lead and zinc deposits mined lie at comparatively shallow depths, the ore bodies are reached by vertical shafts, either single or double compartment. The dimensions of the one-compartment shafts

vary usually from 4 by 5 feet to 5 by 7 feet in the clear, whereas those of two-compartment shafts vary from 5 by 10 feet to 7 by 12 feet. Recently a few three-compartment shafts have been sunk.

The sinking of relatively shallow shafts, 40 to 100 feet deep, is usually a simple matter, but in the deeper shafts, particularly when a large flow of water is tapped, more or less trouble may arise. Difficulties are especially common in the Miami district when sinking to the lower ore-bearing levels, where the underground flow is considerable, and large pumps are necessary. It is believed, however, that as new shafts are sunk and new mines opened such trouble from underground water will be reduced to a minimum.

The shafts are usually sunk according to the American center-cut system and are 6 to 12 feet deeper than the mine level, to allow room for a drainage sump from which the mine water is pumped to the surface.

The amount of cribbing used in the shaft depends upon the character of the ground penetrated and the amount of water entering the shaft from lateral water courses. The shafts in soft-ground mines are usually lined from the collar to the mine level with close cribbing to prevent pieces of rock falling down the shaft and to keep out as much water as possible. Where the ground is more or less hard and the walls of the shaft are fairly firm, the timbers are placed one above the other, an open space alternating with each timber. The cut timbers are usually 2 by 4 inches or 2 by 6 inches in size. The lining is usually "laced," which helps to support it, keeps the shaft free from water and small pieces of rock, and gives the sides a smooth surface.

In the sheet-ground mines the shaft lining usually extends only from the collar of the shaft through the soil and less firm ground near the surface to the "hard ground" beds of rock.

Usually the main shaft is connected directly with the main hopper of the mill. When two or more shafts are used the distance between them is about 300 feet.

METHODS OF RAISING AND SINKING SHAFTS.

At a few mines raising a shaft has been found practicable. A cable attached to a small headframe at the surface, passed through a drill hole to the underground workings, supports a platform. This platform is raised and lowered by means of the cable, and is used to support the air drill and the drill man. After a round of holes has been drilled, the holes are charged and blasted, the platform being removed before the charge explodes. By this method the face is always clear of broken rock. It has been found that the cost of raising a shaft in sheet ground is less than that of sinking to the same depth.



MINING "SHEET-GROUND" ORE WITH LOW FACE. SHOWS IRREGULARITY OF FACE, ALSO BANDING OF ORE.



4. HIGH WORKING FACE IN HARD GROUND. NOTE ABSENCE OF TIMBERING.



B. PILLAR LEFT TO SUPPORT ROOF OF RELATIVELY HIGH WORKING FACE.

The cost of sinking shafts in this district varies according to the nature of the ground, the amount of water encountered, and the size and depth of the shaft. The average cost of sinking to about 200 feet can be figured at \$10 to \$30 per foot.

The actual cost of raising a shaft in sheet ground 210 feet, the size of shaft being 6 by 10 feet in the clear, is shown in Table 6.

TABLE 6.—*Cost of raising shaft in sheet ground.*

[Size of shaft 6 by 10 feet in the clear; depth 210 feet.]

Small headframe for raising and lowering cable:		
Supplies.....	\$51.96	
Labor.....	58.51	
		\$110.47
Power machinery:		
Hoist.....	45.00	
Cable.....	33.14	
Supplies.....	65.33	
		143.47
Drills:		
Drills.....	274.94	
Steel.....	218.87	
		493.81
Explosives:		
Powder.....	454.68	
Exploders.....	53.59	
Fuses.....	39.82	
		548.09
Labor raising:		
Top.....	222.85	
Drilling.....	431.25	
Shoveling.....	62.97	
		717.07
Labor sinking:		
Top.....	126.62	
Drilling.....	255.25	
		381.87
Timbering:		
Cribbing.....	457.13	
Labor.....	59.01	
		516.14
Compressed air.....		350.00
Total cost.....		3,260.92
Cost per foot.....		15.53

In contrast to the figures above, a shaft was previously sunk in similar ground at the same mine to a depth of 240 feet at a total cost of about \$5,000, or \$20.83 per foot. The difference in cost per foot between the two shafts is the more marked because the deeper shaft was the smaller, measuring 5 by 10 feet in the clear, and at the time this shaft was sunk the average wage was about 50 cents lower per shift than at the time the other shaft was raised.

DRILLING.

As the chief object in mining is to excavate the most ore at the least cost, certain conditions in drilling the ore must be considered in order to obtain the best effect in subsequent blasting. First among these are the character of the rock in which the ore bodies lie—whether hard, soft, brittle, or tough—and the form of the ore deposit, whether massive or in stratified layers. Some rocks are creviced and others contain large cavities, variations that make it necessary for the drill man to use his best judgment in placing the holes so that the effect of the explosive in breaking ground will be greatest. Too much emphasis, therefore, can not be placed upon the need of ascertaining in advance, as nearly as possible, the nature of the rock to be mined. Failure to ascertain it is apt to make drilling and blasting costs unnecessarily high.

TYPES OF DRILLS USED.

Several makes of machine drills are used in this district, but practically all are percussion drills driven by compressed air. The air leaves the compressor at a pressure of 90 to 100 pounds, the pressure at the rock drill being about 5 to 10 per cent less because of leakage and other losses in transmission. At some of the mines the air compressors are overtaxed in order that as many drills as possible may be used, thus reducing still more the effective air pressure.

CHARACTER OF ROCK.

The rock drilled, chiefly flint, is hard and brittle and rapidly wears the cutting edge of the bit, but because of its brittleness it is more easily fractured by the concussion of the drill. In some of the mines the gangue rock is tough, so that the footage per shift is less, whereas in other deposits it is more or less soft and the footage for the same amount of time consumed in drilling is much greater than the average. The average footage for drilling in sheet ground is 20 to 50 feet per 8-hour shift for each drill, varying with the character of the ground and the skill of the drill man. Owing to the abrasive action of the flint on the cutting edges of bits, changes every 2 feet are often necessary, and therefore the proper tempering of the steel when bits are sharpened is of much importance.

DEPTH OF HOLES.

The length of the holes drilled varies with the height of the working face and the method of mining. Where the face is about 10 feet high, and no stoping is done, the holes are 8 to 10 feet long. When underhand stoping or benching is used the holes for advancing the heading are usually 6 to 7 feet long, whereas those drilled horizontally

in the underlying bench, which is called the "stope," are usually 12 to 14 and sometimes 16 to 18 feet long. The average diameter of the drill holes is about 3 to 3½ inches at the mouth and 1½ to 1¾ inches at the bottom, depending on the length of the hole. Each drill is usually run by a drill man, known as a machine man, and one helper.

ROCK DUST AND METHODS OF ABATEMENT.

In drilling horizontal holes in dry ground, especially in sheet ground, a large proportion of the cuttings and finely powdered dust gradually passes out of the hole at the mouth. Before any "squibbing" or blasting is done, these cuttings must necessarily be removed. Until recently this was generally accomplished by inserting a long iron pipe about half an inch in diameter, connected with the air hose, into the drill hole. When the air under pressure was turned on, this fine flinty dust was blown from the hole into the drift, where the miners inhaled it. Drilling and blowing dry holes has been one of the chief causes of lung diseases among the miners of the Joplin district. It is believed that much of what is known as "miners' consumption" is caused by sharp particles of siliceous dust, especially flint, being embedded in the lung tissue, forming small receptacles, and that the lungs when affected in this way are much more susceptible to the germs of tuberculosis and other diseases. In view of the particularly harmful effect of the rock dust in the sheet-ground mines, a cooperative investigation was carried on in the Joplin district by the Federal Bureau of Mines, the United States Public Health Service, the State mine inspectors, and the mine operators.

The investigation was conducted by A. J. Lanza, passed assistant surgeon of the Bureau of Public Health Service, and Edwin Higgins, formerly of the Bureau of Mines, under the supervision of G. S. Rice, chief mining engineer of the Bureau of Mines. A survey was made of the conditions in the mines with special reference to rock dust produced in the sheet-ground mines by drilling, blasting, and shoveling. The results of these investigations were published shortly afterwards, in the form of a preliminary report.^a

Following this investigation the State mine inspectors recommended certain regulations and laws to be passed for the abatement of unnecessary dust in the mines and of other insanitary conditions affecting the health of the miners. A final report of the investigation has since been published by the Bureau of Mines as Bulletin 132.^b

Owing to the more stringent regulations adopted and the careful and efficient inspection of the mines by the State officials; the drill

^a Lanza, A. J., and Higgins, Edwin, *Pulmonary disease among miners in the Joplin district, Missouri*, a preliminary report: Technical Paper 105, Bureau of Mines, 1915, 48 pp.

^b Higgins, Edwin, Lanza, A. J., Laney, F. B., and Rice, G. S., *Siliceous dust in relation to pulmonary disease among miners in the Joplin district, Missouri*: Bull. 132, Bureau of Mines, 1917, 112 pp.

cuttings are now removed from the holes by the use of a water hose, which is also used for spraying down the faces and broken ground, thus reducing to a minimum the rock dust suspended in the air. The State mine inspectors at the time of this investigation were Ira L. Burch, Walter W. Holmes, and C. M. Harlan.

PLACING OF HOLES.

To insure efficiency in blasting careful attention is necessary in drilling and placing the holes. The miner or machine man should be familiar with the nature of the ground, in order to arrange his holes to assure the most effective results. He should ascertain the direction of the line of least resistance and drill the holes in such a direction that the charge will break most effectively with the minimum amount of powder that part of the face which he wishes to remove. Other important factors that should be considered in breaking ground are the irregularities of the face and the form in which it will be left after the round of shots is fired. The mine foreman should constantly try to foresee the condition of the face after shooting, and should have the machine men follow, as far as practicable, a system that leaves the face most accessible for subsequent drilling and blasting. As the use of explosives is one of the most important items of cost in hard-ground mining in this district, their use to the best advantage should always be considered. These considerations are followed in many of the mines throughout the district.

BLASTING.

The effectiveness or efficiency of an explosive is usually measured by the quantity of rock broken and the quality of the product. The gases generated by the shot exert pressure in every direction, and the result will depend on the character of the rock in which the explosive is used, the kind of explosive, the manner in which it is used, and the method by which it is exploded.

CHARACTER OF EXPLOSIVES USED.

The strength of the explosive most used in this region is equivalent to that of a 35 to 40 per cent "straight" nitroglycerin dynamite. The "straight" nitroglycerin dynamites are quick acting and are said to have a greater disruptive force than other commercial types of explosives.^a

In wet holes gelatin dynamite is used, as it is relatively impervious to water, whereas for dry holes the low-freezing ammonia dynamites are much used. The ammonia dynamites have a lower rate of deto-

^a Munroe, C. E., and Hall, Clarence, A primer on explosives for metal miners and quarrymen: Bull. 80, Bureau of Mines, 1915, p. 22.

nation and their action is more a pushing and less a shattering. They have the disadvantage, however, of taking up moisture, and consequently must be kept in a dry place.

STORAGE OF EXPLOSIVES.

Explosives reach the mines in 50-pound boxes containing 80 to 140 cartridges, according to the kind of explosive used. The usual supply kept at the mines is 15 to 40 boxes; these are stored on the surface in a magazine situated about 300 feet from the mill and other buildings. The magazines are generally built of wood with double walls, the space between the walls being filled with tailings or manure, and many are surrounded on three sides by tailings. These buildings are used not only as magazines, but as thawing houses, and are equipped with steam pipes, or other means of heating, to keep the dynamite at a temperature above its freezing point. Some of the mines have small magazines underground but these are used only for the storage of fuse and detonators and for the day's supply of explosive. These are situated away from the main drifts and working faces.

LOADING THE HOLES.

The cartridges are prepared and are loaded into the holes by a "powder man," who usually has a helper. Each stick of dynamite is sliced down the side and is loaded separately into the hole with a pointed wooden bar. Chambering of holes or "squibbing," preliminary to inserting the final charge, and the firing of one or two sticks in "blocked" holes have in the past usually been done by the machine men at noon or during the day, but new regulations require this to be done when the men come off shift, in order that the amount of dust thrown into the mine air while miners are at work may be minimized.

After a sufficient charge has been loaded into the hole, the primer or cartridge containing the cap (detonator) and fuse is inserted. Three different ways in which the cap is inserted in the primer are shown in figure 4.^a The primer is usually followed by one or two sticks of dynamite which are tamped with a wooden bar. At a few of the mines specially prepared sticks of stemming^b material are used, whereas, at others no stemming is used. The men as a

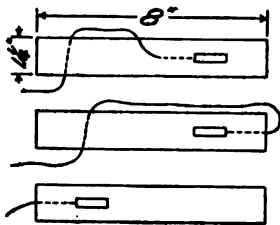


FIGURE 4.—Three ways of inserting detonator in primer as practiced in Joplin district.

^a The Bureau of Mines recommends that the detonator be inserted in the end of the primer as in the bottom diagram in figure 4. To lace fuse through a cartridge of high explosive is dangerous because the explosive, ignited by side spitting of the fuse, may burn before exploding, with the result that poisonous gases will be produced. Detonator should not be inserted so far in the cartridge as is indicated in figure 4.

^b In the publications of the Bureau of Mines the word "stemming" is used to designate the material placed on a charge of explosive in a drill hole, and the word "tamping" to designate the act of placing and ramming the material.

rule do not care to use specially prepared stemming, claiming that its use produces no better results and involves a loss of time. This opinion seems hardly based on proof, in view of the limited amount of experimenting done with explosives in this district; for this reason some of the results of tests made at the Pittsburgh station of the Bureau of Mines* to determine the advantages from the use of stemming are given below.

ADVANTAGES OF USING STEMMING.

In carrying out these tests, the Trauzl lead block—a lead cylinder 20 cm. ($7\frac{7}{8}$ inches) high and 20 cm. ($7\frac{7}{8}$ inches) in diameter, with a bore hole 2.5 cm. (1 inch) in diameter and 12.5 cm. (5 inches) long—was used, as it furnishes one of the simplest means for measuring the relative strength of high explosives.

A known amount of the explosive under test is placed in the bore hole and then fired. The pressure of the gases evolved by the explosive expands the hole into a pear-shaped cavity. The original volume of the bore hole is determined by measuring the amount of water required to fill it exactly; the volume, after a test, is determined in the same way. The difference between the volume of the hole after the firing of the charge and the volume before firing is taken as a measure of the force exerted by the explosive.

The different kinds of stemming used in the tests were dry sand, both tamped and untamped; dry fire clay, tamped and untamped; moist sand, tamped; and moist fire clay, tamped. The table below shows the effect of using different weights and kinds of stemming.

TABLE 7.—*Expansion of bore hole in lead blocks by 20 grams of explosive with different kinds and weights of stemming.*

FORTY PER CENT STRENGTH AMMONIA DYNAMITE.

Kind of stemming.	Weight of stemming, grams.						
	0	6.25	12.5	25	50	100	200
Untamped dry sand..... expansion of bore hole, c. c. .	230	304	359	408	430	434	446
Untamped dry fire clay..... do.....	280	288	347	385	387	381	378
Tamped dry sand..... do.....	230		355	379	383	387	417
Tamped dry fire clay..... do.....	230	286	339	383	389	379	381
Tamped moist sand..... do.....	230	265	293	375	389	419	470
Tamped moist fire clay..... do.....	230	280	341	382	407	462	486

FORTY PER CENT "STRAIGHT" NITROGLYCERIN DYNAMITE.

Untamped dry sand..... expansion of bore hole, c. c. .	367	462	504	569	587	636	642
Untamped dry fire clay..... do.....	367	460	496	574	578	590	589
Tamped dry sand..... do.....	367	454	503	591	592	615	612
Tamped dry fire clay..... do.....	367	385	504	601	602	629	638
Tamped moist sand..... do.....	367	465	525	603	645	697	696
Tamped moist fire clay..... do.....	367	507	547	623	643	696	710

* Snelling, W. O., and Hall, Clarence, The effect of stemming on the efficiency of explosives: Tech. Paper 17, Bureau of Mines, 1915, p. 20.

According to Snelling and Hall* the results obtained from the experiments prove definitely that confinement by the use of stemming greatly increases the efficiency of a charge of explosive.

As the expansion of the bore hole of the lead blocks in the experiments is a good measure of the pressure that is exerted on the wall of a drill hole by the same explosives in actual mining operations, it is reasonable to assume that the useful work done by the explosives would be in the same ratio. The results of the tests show that with quick-acting explosives, such as "straight" dynamite, a small quantity of stemming greatly increases the efficiency, whereas a large quantity is required with slow-burning explosives, such as black blasting powder, for effective results. In addition, the use of stemming reduces the evolution of poisonous gases to a minimum.

FIRING OF SHOTS.

In general, the holes are drilled and "squibbed" on one day, and the loading and blasting are done on the next day. The machine men, as a rule, light their own shots after all other men are out of the mine. The time of shooting is about 4 p. m., or between 12 and 1 o'clock at night, if there are two underground shifts.

The quantity of explosive used per ton of ore broken varies from 1 to 1.5 pounds in hard ground and from 0.2 to 0.6 pound in soft ground, depending on the nature of the ground. To reduce misfires and, consequently, to lessen the quantity of powder used per ton of rock broken, high-grade detonators are recommended. With stronger detonators and lower-grade dynamite it is possible to obtain, in most instances, as good results as with weaker detonators and higher-grade dynamite. As the lower-grade powders are cheaper, their use with the more efficient detonators should reduce the cost per ton of rock broken. The lower-grade dynamite also has a less shattering effect, thus reducing the amount of fines produced before the ore reaches the mill and making possible a saving of mineral that might otherwise be lost as slimes.

The actual quantities of dynamite used per ton of ore broken in three different mines are shown in Table 9.

* Snelling, W. O., and Hall, Clarence, The effect of stemming on the efficiency of explosives: Tech. Paper 17, Bureau of Mines, 1915, p. 15.

TABLE 8.—*Actual quantities of dynamite used per ton of ore broken in three mines.*

Year.....	Quantity of dynamite used per ton of ore.			Remarks.
	1914	1915	1916	
	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	
Mine 1.....	1.417	1.395	1.326	Sheet-ground, hard, tough, face 12 to 15 feet high.
Mine 2.....	1.655	.784	.874	Sheet-ground, hard; 1914 included heading, 1915 and 1916 stope only (taking up floor of mine).
Mine 3.....	1.056	1.326	1.261	Hard ground, Miami district, near Commerce, high face.

SHOVELING AND UNDERGROUND TRAMMING.

After the ground has been broken the ore is shoveled into cars or cylindrical tubs which are trammed by the shovelers or are hauled on a small truck by mules from a lay-by (siding) to the shaft. The tubs, locally known as "cans," usually measure 30 by 30 inches in diameter and hold about 1,000 pounds of broken ore. A piece of sheet iron or a small floor of wooden planks is laid so as to provide a flat and smooth surface from which to shovel. The bowlders and larger pieces of rock are lifted into the tub, those too large for one man to handle being broken with sledges or blasted. The blasting of bowlders is known as "boulder popping." The shovelers are good workmen, as a rule, averaging about 20 tons per man per shift. They are usually paid by the car or tub. This system incites the men to work hard and tends to increase the tonnage, but has the serious disadvantage that the men are apt not to fill the tubs full in order to get a greater number to their credit; this drawback can be overcome only by careful supervision. As a result of this tendency of the men and of the fact that the cars or tubs are rarely weighed, the reported tonnage of ore hoisted is apt to be somewhat higher than the actual quantity. If the ore hoisted were weighed, the men could be paid by the ton, or could receive a daily wage combined with a bonus for extra tonnage above a required amount.

Besides the usual underground haulage system already mentioned, both gasoline and electric motors are used in some of the larger mines with satisfactory results. (See Pl. IX, A.) At one or two mines a system of rope haulage, driven by electric motors, is used to good advantage. (See Pl. IX, B.) Traction motors, where used, necessitate a wider track gage and heavier rails.

TIMBERING.

The main object of timbering in mines is not to support the total weight of the overlying strata but to keep the roof in place and prevent falls of rock and ore. (See Pl. X, A.) Cracking or gradual



A. GASOLINE MOTOR USED FOR UNDERGROUND HAULAGE



B. ROPE HAULAGE USED UNDERGROUND.



A. CAVE-IN RESULTING FROM PILLARS GIVING WAY.



B. REINFORCED-CONCRETE PILLAR IN COURSE OF CONSTRUCTION.

breaking of timbers warns the workmen to place additional timbering or to escape before a fall occurs. In the best practice the timbers are so placed that the pressure is evenly distributed and in line with the thrust that they are to resist. The joints are so made that the pressure, up to the crushing strength of the timber, tends to hold the structure together rather than to weaken it.

In the mines of the Joplin district, timbering is rarely used in either the sheet-ground or the hard-ground deposits, for the roof is generally firm and pillars are left at intervals of 25 to 60 feet, center to center, according to the nature of the ground and the height of the roof above the floor. Posts are used to support dangerous slabs of rock, and in some of the mines cribs filled with waste rock are used as roof supports.

In the soft-ground deposits, however, timbering is largely employed. These deposits, as already mentioned, are mined by lateral drifts in the ore body, the roof and walls being supported by timbering or by a combination of pillars and timbering. Both fore-poling and square-set systems are used. Where the drifts are narrow and the height of the face is relatively low, fore-poling is used to good advantage, the square-set system being more applicable to fairly wide drifts and relatively high faces of ore. The drifts are seldom timbered more than two square sets high. In a few mines reinforced concrete pillars (Pl. X, *B*) have been used as supports, especially at the shaft, with more or less satisfactory results. The concrete mixture is poured through a drill hole from the surface into a form placed in the underground workings just below the drill hole.

Plate XI shows the general methods of timbering used in the district.

PUMPING.

The quantity of water pumped from the mines in the Joplin district varies from about 200 to 2,000 gallons a minute. The working levels of the sheet-ground mines are as a rule fairly dry, whereas in many of the soft-ground mines much water is encountered. The water usually enters the mine through watercourses or by intermittent seepage, and unless the workings have a natural drainage outlet, it has to be pumped out. Seepage in the mines is rather general throughout the district, as the ore deposits are flat and relatively shallow. At a few of the mines underground watercourses have necessitated heavy pumping, especially in the Miami field, and in several instances have caused much difficulty in the sinking of shafts. The usual practice is to drain the underground water to a sump at the foot of a shaft, either by means of ditches along the drifts or by some natural drainage course, and then pump it to the surface.

Mine pumps commonly used in this district include the usual piston types driven by steam and electrically driven centrifugal pumps, their size being governed by the flow of water to be handled. In a few places in the district, pumping is on a cooperative basis, the water from several mines being handled by one pumping station and each mine paying an apportioned sum to the company doing the pumping.

Because of the mines being shallow, and the topography having slight relief, a heavy rainfall greatly increases the amount of water that enters the workings; and if the pumping equipment at such times is insufficient the mine is shut down temporarily, causing loss of both time and money.

HOISTING AND SURFACE TRAMMING.

Direct winding without any attempt to balance the load is the usual practice, the tub or skip being hoisted by power and lowered by gravity. The hoisting engine is coupled or geared direct to the shaft of the drum, which is provided with friction devices or positive clutches or brakes. Steam is the usual power, though electric hoists are in use at many mines. Cars, if used, are placed in cages and hoisted in the usual way. When a tub has been hoisted to the top of the headframe the hoistman covers the opening with a lid to prevent any material from falling down the shaft, and then dumps the load on a chute leading directly to the grizzly over the hopper. Cages and self-dumping skips of 1 to 3 tons capacity, provided with shaft guides and safety catches, are used at a few mines.

Connected with each shaft is a storage bin or hopper, and with few exceptions the mill hopper is connected directly with the main shaft. The ore hoppers have a capacity of 100 to 500 tons each, although at some mines the capacity of the mill hopper is more than 500 tons in order to hold ore enough to supply the mill during an extra shift. If hoisting is done from more than one shaft, the ore is trammed to the mill hopper over inclined or horizontal tramways, or, where the distance is too great, by aerial tramways or surface motor haulage. The capacity of the surface tram cars is 1 to 3 tons.

POWER.

Electricity, natural gas, coal, and oil are all used in the district as sources of power. Natural gas seems to be most favored, owing to its cheapness; coal and oil the least. However, during the winter months when the gas pressure becomes low, coal is used to a large extent. At many of the properties the mill is driven by electric motors or gas engines and the hoisting and compressor engines are driven by steam, whereas at others electric power is used throughout.



A. FOREPOLING IN SOFT GROUND.



B. METHOD OF TIMBERING IN SOFT-GROUND MINING.

ORE-DRESSING PRACTICE.

In general, the average lead and zinc content of the ores mined and is low as compared with that of other lead and zinc fields throughout the United States. Owing to this fact and to the high of concentrate produced, the ratio of concentration is unusually high.

OF CONCENTRATE AND PERCENTAGE OF RECOVERY.

RESULTS OF MILL TESTS.

The object of most of the operating companies has been to produce a high grade of concentrate as possible in order to receive the highest base price for their product, a practice that has resulted in lowering the percentage of zinc recovered. The average percentage of recovery is 60 to 70 per cent; or, in other words, for every 2 tons of zinc concentrate produced 1 ton is lost. The percentages of recovery from actual mill tests are shown in Tables 9 to 11, on pages 44 to 52.

The concentration of the ore is commonly effected by crushing to a fine size and roughing and cleaning the crushed material over two large Cooley jigs of the Harz type, and running the finer material over the usual types of sand and slime tables. The treatment of the ores by flotation is being tried at several of the larger mills. The various steps in the concentration of the ores in this district are shown more fully by the typical flow sheets in figures 5 to 8 and Plate XII. The significance of the numbers on each figure are explained by the tabulation opposite the figure.

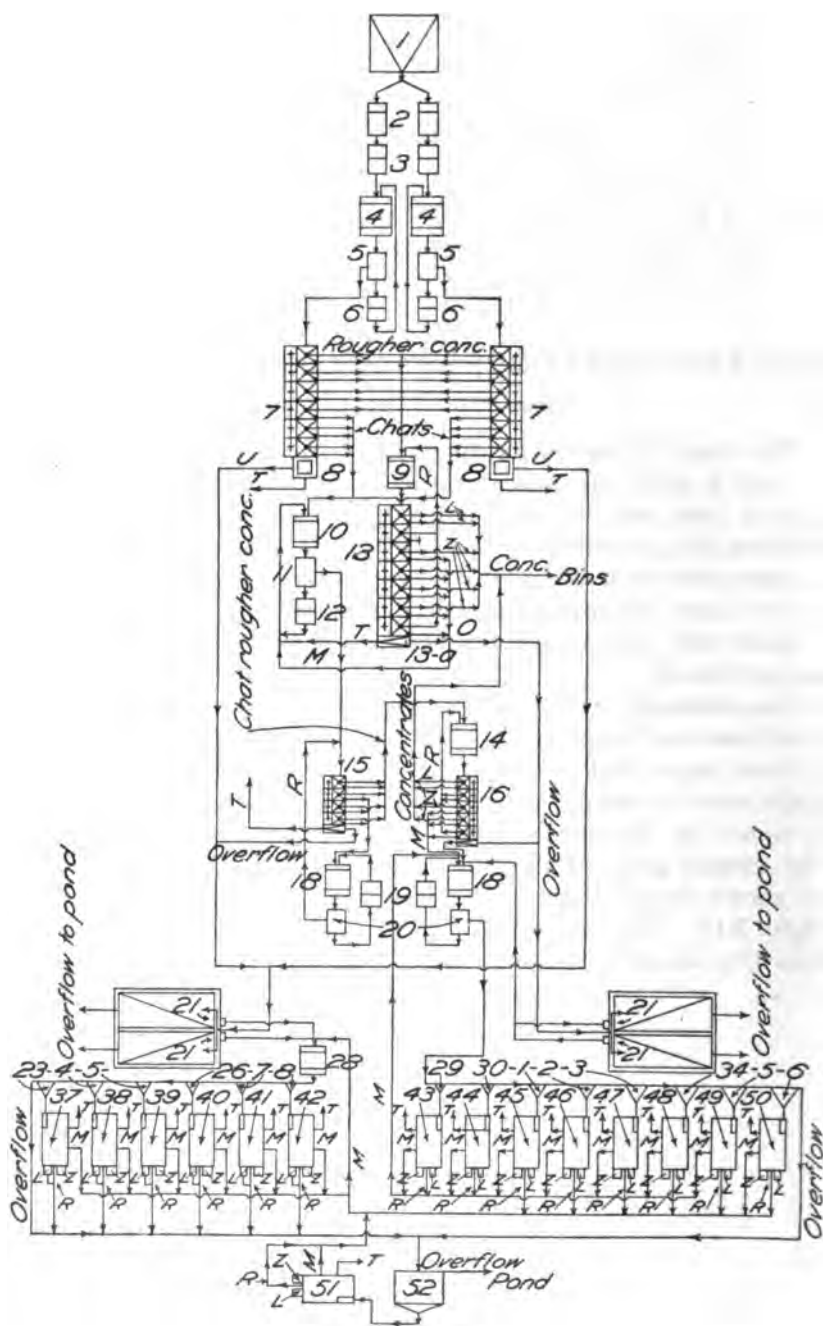


FIGURE 5.—Flow sheet of mill in Oronogo district, Missouri. L, lead concentrate; Z, zinc concentrate; M, middling; O, overflow; R, return; T, tailing. For significance of numbers see table on opposite page.

Data on parts shown in figure 5.

1. Hopper, 400-ton capacity.
2. Crushers (two), 18 by 12 inches.
3. Rolls (two), 36 by 14 inches.
4. Elevators (two).
5. Trommel screens (two), 8 by 4 feet, $\frac{1}{4}$ -inch openings.
6. Rolls (two), 30 inches in diameter.
7. Rougher jigs (two), cells 36 by 48 inches.
8. Dewatering screens (two).
- 9, 10. Elevators.
11. Trommel screen, 8 by 4 feet, $\frac{1}{4}$ -inch openings.
12. Rolls, 36 inches in diameter.
13. Cleaner jig, cells 30 by 42 inches.
- 13a. Dewatering box.
14. Elevator.
15. Rougher chat jig, cells 30 by 42 inches, with dewatering box.
16. Cleaner chat jig, cells 26 by 36 inches, with dewatering box.
18. Elevators (two).
19. Rolls (two sets), 24 inches in diameter.
20. Trommel screens (two), 36 by 72 inches, size of openings 3 mm.
21. Settling tanks.
22. Elevator.
- 23-36. Hydraulic classifiers.
- 37-51. Tables.
52. Tank.

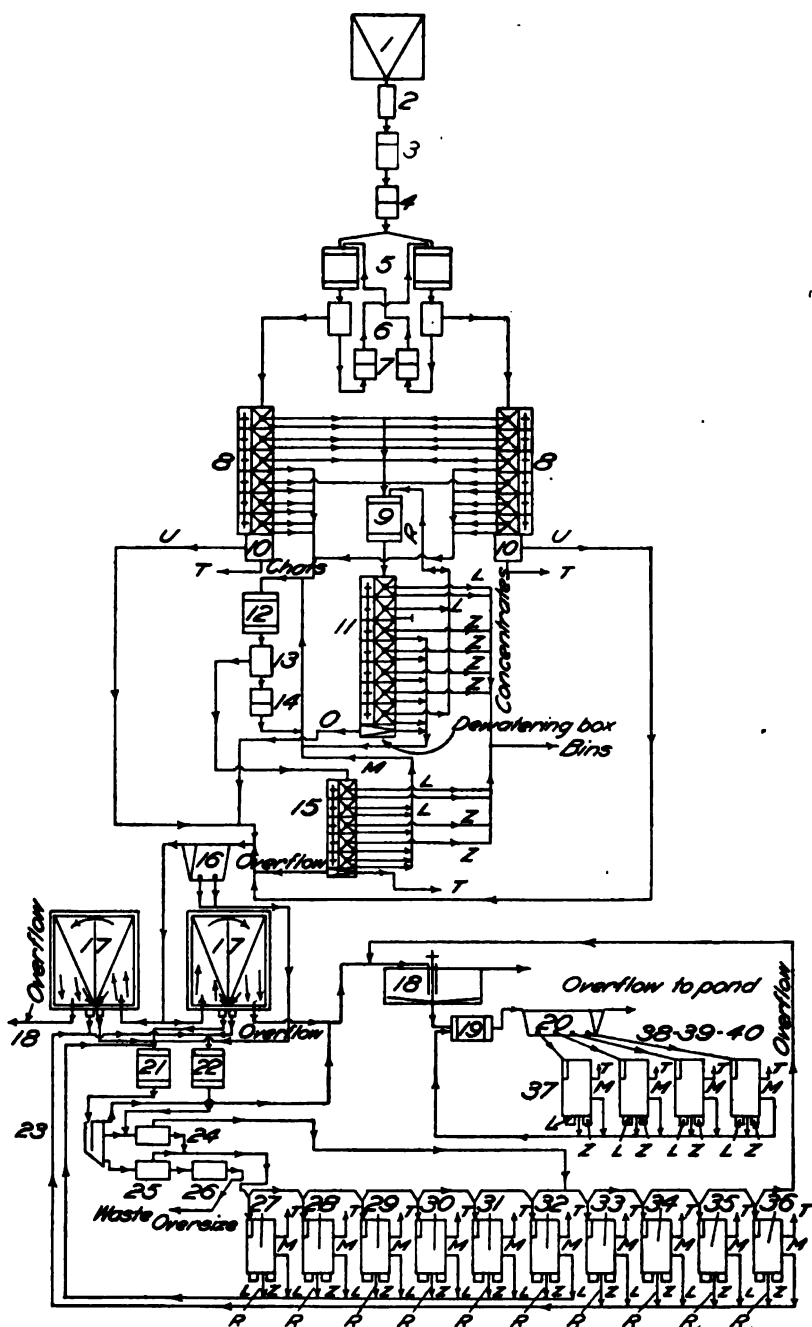


FIGURE 6.—Flow sheet of mill in west Joplin district, Missouri. L, lead concentrate; Z, zinc concentrate; M, middling; O, overflow; T, tailing; R, return. For significance of numbers see table on opposite page.

Data on parts shown in figure 6.

1. Hopper, capacity 400 tons.
2. Shaking screen, 20 inches by 4 feet 6 inches, with $\frac{1}{2}$ -inch openings.
3. Crusher, 18 by 12 inches, speed 390 r. p. m.
4. Rolls, 42 by 16 inches, speed 27 r. p. m.
5. Elevators (two), 20 inches, buckets spaced 10 inches, speed 275 feet per minute.
6. Trommels (two), 4 by 12 feet, $\frac{1}{2}$ -inch openings, speed 25 r. p. m.
7. Return rolls (two), 36 by 14 inches, speed 30 r. p. m.
8. Rougher jigs (two), Cooley type, 6 cells, 36 by 48 inches, speed 110 strokes per minute.
9. "Smitem" elevator, 16-inch, 320 feet per minute.
10. Dewatering trommels (two), 5 by 5 feet, 2-mm. screen openings, speed $1\frac{1}{2}$ r. p. m.
11. Cleaner jig, Cooley type, 7 cells 32 by 42 inches, speed 180 strokes per minute.
12. Chat elevator, 16-inch, speed 300 feet per minute.
13. Trommel, 4 by 8 feet, screen opening $\frac{1}{2}$ -inch, speed 26 r. p. m.
14. Chat rolls, 30 by 14 inches, speed 56 r. p. m. (spaced rolls).
15. Chat jig, Cooley type, 5 cells 32 by 42 inches, speed 180 strokes per minute.
16. V tank, used to eliminate large proportion from feed to settling tanks.
17. Settling tanks (two), double tanks, 30 by 40 feet (plan view), 8 feet deep at discharge, 2 feet deep at back end near partition.
18. Dorr thickener, 40 by 8 feet, speed 10 revolutions per hour.
19. Slime elevator, 14-inch, 350 feet per minute.
20. V tank, discharge to slime tables, overflow to pond.
21. Coarse-sand elevator, receiving sand from first half of settling tank, 14-inch, speed 350 feet per minute.
22. Fine-sand elevator, receiving sand from second half of settling tank, 14-inch, speed 350 feet per minute.
23. Bird classifier, discharge products to trommels, overflow to Dorr thickener.
24. Trommel, 3 by 4 feet, speed 14 r. p. m., $\frac{1}{2}$ -mm. screen opening, undersize to classifier above table 33 (seventh table).
25. Trommel, 4 by 8 feet, speed 14 r. p. m., $1\frac{1}{2}$ -mm. screen opening, receiving oversize from trommel 24 and coarse discharge from classifier 23. Undersize to tables.
26. Trommel, same as No. 25, receives oversize from trommel 25. Oversize goes to waste as tailing, undersize to table feed.
- 27-32. Concentrating tables, feed consisting of undersize from trommels 25 and 26 and oversize from trommel 24.
- 33-36. Concentrating tables, feed consisting of undersize from trommel 24 and overflow from classifier to table 33. Overflow from classifier to table 36 goes to Dorr thickener.
- 37-40. Concentrating tables, receiving spigot discharge from V tank 20.

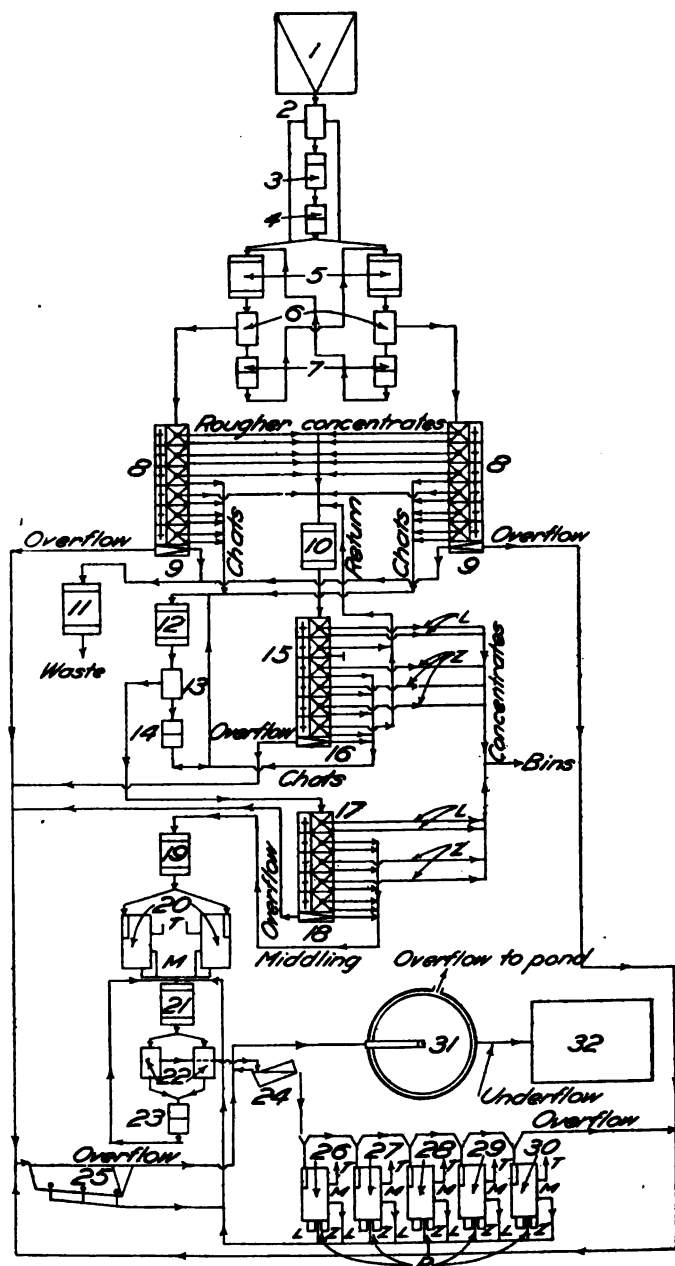


FIGURE 7. Flow sheet of mill in Granby district, Missouri. L, lead concentrates; Z, zinc concentrates; M, middling; T, tailing; R, return. For significance of numbers see table on opposite page.

Data on parts shown in figure 7.

1. Mill storage bin, 400-ton capacity, all ore through 6-inch grizzly.
2. Shaker screen, $\frac{1}{8}$ -inch plate with $1\frac{1}{8}$ -inch openings, stroke $1\frac{1}{4}$ inches, speed 125 strokes per minute.
3. Crusher, Blake type, size 24 by 12 inches, speed 350 r. p. m., set for $1\frac{1}{2}$ -inch opening.
4. Cornish rolls, size 42 by 16 inches, geared, speed 30 r. p. m.
5. Feed elevators (two), 20-inch belt, buckets spaced 20 inches apart, belt speed 300 feet per minute.
6. Trommels (two), size 48 by 96 inches, slope 1 inch per foot, speed 28 r. p. m. Screen is $\frac{3}{8}$ -inch plate with $\frac{3}{8}$ -inch round, punched holes.
7. Cornish rolls (two), size 36 by 14 inches, geared, speed 36 r. p. m.
8. Cooley jigs (two), Harz type, six cells each 36 by 48 inches in size. Screens (grates) are punched plate with slotted holes. Sizes of screen openings: Cells 1 to 3, $\frac{3}{8}$ -inch, cells 4 and 5, $\frac{1}{4}$ -inch, cell 6, $\frac{1}{8}$ -inch. Lengths of plunger stroke for cells 1 to 6 are $1\frac{1}{4}$, $1\frac{1}{4}$, $1\frac{5}{8}$, $1\frac{1}{2}$, $1\frac{3}{4}$, $1\frac{1}{8}$ inches, respectively. Speed 110 strokes per minute.
9. Dewatering box.
10. Elevator ("smitem"), 16-inch belt, speed 400 feet per minute.
11. Elevator (tailings), 20-inch belt, belt speed 420 feet per minute.
12. Elevator (chats), 16-inch belt, belt speed 400 feet per minute.
13. Trommel, size 48 by 72 inches, $\frac{1}{8}$ -inch round holes in screen, speed 28 r. p. m.
14. Rolls (chats), Cornish type, geared, size 30 by 14 inches.
15. Cooley jig, Harz type, with six cells each 36 by 42 inches in size. Screens (grates) are punched plate with slotted holes. Sizes of screen openings: Cell 1, $\frac{1}{4}$ inch, cells 2 to 5, $\frac{1}{8}$ inch, cell 6, $\frac{1}{16}$ inch. Lengths of plunger stroke for cells 1 to 6 are $\frac{1}{4}$, $\frac{1}{4}$, $\frac{5}{8}$, $\frac{5}{8}$, $\frac{5}{8}$, $\frac{5}{8}$ inch, respectively. Speed 165 strokes per minute.
16. Dewatering box.
17. Cooley jig, Harz type, with five cells each 36 by 42 inches in size. Screens (grates) are punched plate with slotted holes. Sizes of screen openings: Cells 1 to 4, $\frac{1}{8}$ -inch, cell 5, $\frac{1}{16}$ -inch. Lengths of plunger stroke for cells 1 to 5 are $\frac{1}{4}$, $\frac{1}{4}$, $\frac{5}{8}$, $\frac{5}{8}$, and $\frac{1}{4}$ inch, respectively. Speed 180 strokes per minute.
18. Dewatering box.
19. Elevator (for "rougher" tables), 16-inch belt, speed 440 feet per minute.
20. Rougher tables (two).
21. Elevator (sand), 20-inch belt, belt speed 440 feet per minute.
22. Trommels (two), size 48 by 96 inches, $1\frac{1}{2}$ -inch round holes in screen. Speed 24 r. p. m.
23. Rolls, size 36 by 14 inches, belted, speed 95 r. p. m., steel shells.
24. Akins classifier, 54 inches in diameter, speed 5 r. p. m., slope $2\frac{1}{2}$ inches per foot.
25. V tanks (three), height 12 feet, length 30 feet, width 12 feet.
- 26, 27, 28. Sand tables, speeds 238 strokes per minute.
- 29, 30. Sand tables, speeds 267 strokes per minute.
31. Dorr thickener, 40 feet in diameter, speed 8.6 revolutions per hour.
32. Flotation plant.

Data on parts shown in figure 8.

1. Hopper.
2. Crushers (two), size 18 by 12 inches, speed 354 r. p. m.
3. Rolls (two sets), Cornish type, size 42 by 16 inches, speed 24.8 r. p. m.
4. Elevators (two), size 24 inches, speed 372 feet per minute.
5. Trommels (two), size 48 by 96 inches, $\frac{3}{8}$ -inch openings, speed 21.2 r. p. m.
6. Rolls (two sets), Cornish type, size 36 by 14 inches, speed 23.6 r. p. m.
7. Dewatering screens (two), 48 by 48 inches, $1\frac{1}{2}$ -inch openings, speed 4 r. p. m.
8. Elevators (two), size 18 inches, speed 310 feet per minute.
9. Hopper, 300-ton capacity.
10. Cooley rougher jigs (two), Harz type, with six cells, each 42 by 48 inches in size, speed 91 strokes per minute.
11. Dewatering boxes (two).
12. Elevator, 16-inch, speed 376 feet per minute.
13. Trommel, size 36 by 72 inches, $\frac{1}{4}$ -inch openings, speed 24 r. p. m.
14. Rolls, size 36 by 14 inches, speed 35 r. p. m.
15. Cooley chat jig, Harz type, five cells, each 36 by 42 inches in size, speed 114 strokes per minute.
16. Elevator, 16-inch, speed 384 feet per minute.
17. Elevator, 24-inch, speed 392 feet per minute.
18. Trommel, size 36 by 72 inches, 2-mm. openings, speed 24.6 r. p. m.
19. Dewatering screen, size 48 by 48 inches, $1\frac{1}{2}$ -inch openings, speed 4 r. p. m.
20. Elevator, 18-inch, speed 373 feet per minute.
21. Rolls, size 36 by 14 inches, speed 93 r. p. m.
22. Cooley cleaner jig, Harz type, 7 cells, size of each cell 36 by 42 inches, speed 181 strokes per minute.
23. Cooley sand jig, Harz type, 7 cells, each 30 by 36 inches in size, speed 187 strokes per minute.
24. Dewatering box.
25. Concentrate bins.
26. Settling tanks (two), size 30 by 22 feet.
- 27, 28. Dorr thickeners (two), 36 feet in diameter, speed 10 r. p. m.
- 29, 30. Elevators.
31. Henry screen, $1\frac{1}{2}$ -inch openings.
- 32 to 35. Slime tables (four).
36. Flotation plant.
37. Elevator.
38. Akins classifier, 54 inches in diameter, speed 6 r. p. m., slope $2\frac{1}{2}$ inches per foot.
- 39, 40. Slime tables (two).
- 41 to 50. Sand tables.
- 51, 52, 53. Elevators.
- 54, 55. Roughing tables.
- 56 to 59. Middling tables.
- 60, 61. Lead tables.

TABLE 9.—Results of concentration tests with mills treating sheet-ground ore.

MILL IN WEBB CITY DISTRICT.

Screen analysis of mill tailings.				Zinc (Zn) content.		
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Cumulative percentage of total zinc.
	Mm.	Inches.				
			<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
3	6.680	0.263	29.47		0.53	20.0
6	3.327	.131	28.38	57.85	.68	44.7
10	1.651	.065	15.48	73.33	.80	60.5
20	.833	.0328	16.10	89.43	.45	69.8
35	.417	.0164	5.50	94.93	.42	72.7
65	.206	.0082	1.37	96.30	.96	74.4
100	.147	.0058	.63	96.93	1.10	75.3
150	.104	.0041	.40	97.33	1.25	75.9
200	.074	.0029	.32	97.65	1.58	76.5
<200	.074	.0029	2.35	100.00	7.90	100.0
Total or average.....			100.00		.78	100.0
Check sample.....					.67	

Ore treated.....	Tons.	
Lead and zinc concentrates produced.....		1,960
Total tailings.....		57.43
Solids in overflow from settling tanks.....		1,902.57
Tailings, not including overflow.....		40.22
Zinc concentrates.....		1,862.35
Zinc (Zn) in—	Percent.	
Zinc concentrates.....	55.0	23.623
Lead concentrates.....	1.2	.174
Tailings without overflow.....	.70	12.478
Overflow.....	7.04	2.831
Total zinc in ore treated.....		39.106
Percentage of zinc recovery.....		60.4

TABLE 9.—Results of concentration tests with mills treating sheet-ground ore—Contd.

MILL IN CARTERVILLE DISTRICT.

Screen analysis of mill tailings not including overflow.					Zinc (Zn).		
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
3	6.680	0.263	Per cent. 33.58	Per cent. 33.58	Per cent. 0.31	24.3	
6	3.327	.121	30.75	64.33	.44	31.6	55.9
10	1.651	.065	16.84	81.17	.44	17.3	73.2
20	.833	.0328	8.78	89.95	.31	6.35	79.55
35	.417	.0164	4.60	94.55	.31	3.3	82.85
65	.208	.0082	2.57	97.12	.55	3.3	86.15
100	.147	.0058	.98	98.10	.57	1.3	87.45
150	.104	.0041	.80	98.90	1.04	1.95	89.40
200	.074	.0029	.21	99.11	1.56	.8	90.2
<200	.074	.0029	.89	100.00	4.68	9.8	100.00
Total or average.....			100.00		.43	100.00	
Check sample.....					.49		

		Tons.
Ore treated.....	3,456	
Concentrates produced (lead and zinc).....	65.53	
Total tailings from mill.....	3,390.47	
Solids in overflow from settling tanks.....	52.86	
Tailings, not including overflow.....	3,337.61	
Zinc concentrates.....	62.765	
Lead concentrates.....	2.765	
Zinc (Zn) in—	Per cent.	
Zinc concentrates.....	61.88	38.839
Lead concentrates.....	15.00	.415
Mill tailings without overflow.....	.49	16.355
Solids from overflow.....	6.38	3.360
Total zinc in ore treated.....		58.969
Percentage of zinc recovery.....	65.86	

TABLE 9.—Results of concentration tests with mills treating sheet-ground ore—Contd.

MILL IN PROSPERITY DISTRICT.

Screen analysis of mill tailings not including overflow.					Zinc (Zn).		
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
3	6.680	0.263	9.90	9.90	0.53	12.3	12.3
6	3.327	.131	38.82	48.72	.37	33.8	46.1
10	1.651	.065	23.76	72.48	.29	16.2	62.3
20	.833	.0328	11.43	83.91	.19	5.1	67.4
35	.417	.0164	6.24	90.15	.25	3.7	71.1
65	.208	.0082	4.12	94.27	.26	2.55	73.65
100	.147	.0058	1.94	96.21	.42	1.9	75.55
150	.104	.0041	1.45	97.66	.95	3.3	78.85
200	.074	.0029	.19	97.85	1.16	.5	79.35
<200	.074	.0029	2.15	100.00	4.07	20.65	100.00
Total or average.....			100.00		.42	100.00	
Check sample.....					.45		

Ore treated.....	Tons.	470
Concentrates produced (zinc concentrates).....	8.545	
Tailings.....	461.455	
Solids in overflow from settlings.....	12.457	
Tailings, not including overflow.....	448.998	
Zinc (Zn) in—	Per cent.	
Concentrates.....	60.18	5.142
Tailings without overflow.....	.45	2.020
Overflow.....	5.25	.654
Total zinc in ore treated.....	7.816	
Percentage of zinc recovery.....	65.79	

TABLE 9.—Results of concentration tests with mills treating sheet-ground ore—Contd.

MILL IN PORTO RICO DISTRICT.

Screen analysis of mill tailings not including overflow.					Zinc (Zn) content.		
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
	13.33	0.535	Per cent.	Per cent.	Per cent.		
3	6.680	.263	6.89	38.97	0.30	5.6	
6	3.327	.131	32.08	65.73	.22	19.3	24.9
10	1.651	.065	26.75	84.12	.20	14.6	39.5
20	.833	.0328	18.40	93.19	.19	9.5	49.0
35	.417	.0164	9.07	95.89	.30	7.4	56.4
65	.208	.0082	2.70	97.69	.58	4.1	60.5
100	.147	.0068	1.80	98.13	1.27	6.3	66.8
150	.104	.0041	.44	98.53	2.98	3.5	70.3
200	.074	.0029	.89	98.73	3.35	3.6	73.9
<200	.074	.0029	.21	98.73	5.70	3.3	77.2
			1.27	100.00	6.55	22.8	100.0
Total or average.....			100.00		.37	100.0	
Check sample.....					.41		

		Tons.
Ore treated	1,994	
Concentrates produced	35.5	
Tailings.....	1,958.50	
Solids in overflow from settling tanks.....	34.73	
Tailings, not including overflow	1,923.77	
Zinc (Zn) in—	Per cent.	
Concentrates.....	59.4	21.087
Tailings without overflow	.41	7.887
Overflow.....	5.01	1.740
Total zinc in ore treated.....		30.714
Percentage of zinc recovery	68.6	

TABLE 9.—Results of concentration tests with mills treating sheet-ground ore—Contd.

MILL IN DUENWEG DISTRICT.

Screen analysis of mill tailings not including overflow.					Zinc (Zn) content.		
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
	12.33	0.525	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>		
3	6.680	.263	18.50		0.60	21.05	
6	3.327	.131	35.93	54.43	.46	31.35	52.40
10	1.651	.085	19.48	73.91	.42	15.5	67.90
20	.833	.0328	12.59	86.50	.36	8.6	76.5
35	.417	.0164	7.29	93.79	.33	5.3	81.8
65	.208	.0082	3.17	96.96	.36	2.15	83.95
100	.147	.0058	.65	97.61	.74	.9	84.85
150	.104	.0041	.54	98.15	.95	.95	85.80
200	.074	.0029	.50	98.65	.84	.8	86.60
<200	.074	.0029	.30	98.95	1.41	.8	87.4
			1.05	100.00	6.30	12.6	100.00
Total or average.....			100.00		.53	100.00	
Check sample.....					.60		

		Tons.
Ore treated.....	938	
Concentrates produced (lead and zinc).....	20.53	
Tailings.....	917.47	
Solids in overflow from settling tanks.....	42.52	
Tailings not including overflow.....	874.95	
Zinc concentrates.....	17.03	
Lead concentrates.....	3.50	
Zinc (Zn) in—	Per cent.	
Zinc concentrates.....	61.47	10.468
Tailings without overflow.....	.60	5.250
Overflow.....	4.21	1.790
Lead concentrates.....	1.00	0.035
Total zinc in ore treated		17.543
Percentage of zinc recovery.....	59.67	

TABLE 10.—Results of concentration test with mills treating hard-ground ore other than sheet-ground.

MILL IN CARL JUNCTION DISTRICT.

Screen analysis of mill tailings not including overflow.					Zinc (Zn) content.		
Mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
	12.33	0.525	Per cent.	Per cent.	Per cent.		
8	6.690	.263	9.20	41.17	0.24	3.15	32.45
6	3.227	.131	31.97	65.18	.64	29.3	48.60
10	1.651	.065	24.01	81.17	.47	16.15	67.30
20	.833	.0328	15.99	90.07	.38	8.7	82.65
25	.417	.0164	8.90	94.77	.42	5.25	87.85
35	.208	.0082	4.70	96.39	.77	5.2	71.45
65	.147	.0058	1.62	97.28	1.55	3.6	73.65
100	.104	.0041	.89	97.67	1.72	2.2	74.95
150	.074	.0029	.39	97.89	2.27	1.3	76.10
200	.074	.0029	.22	100.00	3.69	1.15	100.0
<200	.074	.0029	2.11		7.90	23.90	
Total or average.....			100.00		.70	100.00	
Check sample.....					.76		

	Tons.
Ore treated.....	1,975
Zinc concentrates produced.....	67.45
Tailings.....	1,907.55
Solids in overflow from settling tanks.....	65.09
Tailings without overflow.....	1,842.46
Zinc (Zn) in—	Per cent.
Concentrates.....	60.1 40.537
Tailings without overflow.....	.76 14.003
Overflow.....	6.95 4.524
Total zinc in ore treated.....	59.064
Percentage of zinc recovery.....	68.63

TABLE 10.—Results of concentration test with mills treating hard-ground ore other than sheet-ground—Continued.

MILL IN COMMERCE, OKLA., DISTRICT.

Screen analysis of mill tailings, not including overflow.					Zinc (Zn) content.		
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
3	6.680	0.263	Per cent. 1.12	Per cent. -----	Per cent. 0.60	0.8	-----
6	3.327	.131	37.60	38.72	.70	30.4	31.2
10	1.661	.066	29.21	67.93	.80	26.95	58.15
20	.833	.0328	10.27	78.20	.90	10.65	68.80
35	.417	.0164	5.10	83.30	1.15	6.8	75.60
65	.208	.0082	3.42	86.72	1.13	4.45	80.05
100	.147	.0058	2.72	89.44	.75	2.35	82.40
150	.104	.0041	2.38	91.82	.75	2.1	84.50
200	.074	.0028	1.54	93.36	.95	1.7	86.20
<200	.074	.0028	6.64	100.00	1.80	13.8	100.00
Total or average.....			100.00	-----	0.86	100.00	-----
Check sample.....				-----	0.85		-----

		Tons.
Ore treated.....		1,504
Concentrates produced.....		100.46
Tailings.....		1,403.54
Solids in overflow from settling tanks.....		122.20
Mill tailings without overflow.....		1,281.34
Zinc concentrates.....		39.65
Lead concentrates.....		60.81
Zinc (Zn) in—	Per cent.	
Zinc concentrates.....	47.03	18.647
Lead concentrates.....	2.09	1.271
Tailings without overflow.....	.85	10.891
Solids from overflow.....	1.13	1.381
Total zinc in ore treated.....		32.190
Percentage of zinc recovery.....	57.93	
Lead (Pb) in—		
Lead concentrates.....	80.93	49.214
Zinc concentrates.....	2.44	0.967
Tailings without overflow.....	0.20	2.563
Overflow.....	0.14	0.171
Total lead in ore treated.....		52.915
Percentage of lead recovery.....	93.06	
Quantity of lead and zinc (metallic) in ore treated.....		85.105
Quantity of available lead and zinc in concentrates.....		67.861
Percentage of lead and zinc recovery combined.....	79.74	

TABLE 10.—Results of concentration test with mills treating hard-ground ore other than sheet-ground—Continued.

MILI. IN COMMERCE, OKLA., DISTRICT.

Screen analysis of mill tailings, not including overflow.					Zinc (Zn) content.		
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
3	6.680	0.263	Per cent. 4.99	Per cent. 0.68	Per cent. 3.0		
6	3.327	.131	29.72	34.71	1.08	28.45	31.45
10	.651	.065	22.98	57.69	1.12	22.85	54.30
20	.333	.0328	14.57	72.26	.85	11.0	65.30
35	.417	.0164	12.50	84.76	.87	9.65	74.95
65	.208	.0082	2.02	86.78	.79	1.4	76.35
100	.147	.0058	1.01	87.79	.79	.7	77.05
150	.104	.0041	.70	88.49	.97	.6	77.65
200	.074	.0029	.50	88.99	1.13	.5	78.15
<200	.074	.0029	11.01	100.00	2.24	21.85	100.00
Total or average.....			100.00		1.13	100.00	
Check sample.....					1.1		

	Tons.	
Ore treated.....	1,498	
Zinc concentrates.....	51.05	
Lead concentrates.....	45.50	
Concentrates produced (lead and zinc).....	96.55	
Tailings.....	1,401.45	
Solids in overflow from settling tanks.....	84.13	
Tailings without overflow.....	1,317.32	
Zinc (Zn) in—	Per cent.	
Zinc concentrates.....	42.80	21.849
Lead concentrates.....	2.50	1.137
Tailings without overflow.....	1.10	14.490
Overflow.....	3.28	2.759
Total zinc in ore treated.....		40.235
Percentage of zinc recovery.....	54.33	
Lead (Pb) in—		
Lead concentrates.....	78.0	34.580
Zinc concentrates.....	2.7	1.378
Tailings without overflow.....	.2	2.635
Overflow.....	1.92	1.615
Total lead in ore treated.....		40.208
Percentage of lead recovery.....	86.00	
Quantity of lead and zinc in ore treated.....		80.443
Quantity of lead and zinc available in concentrates.....		56.429
Percentage of lead and zinc recovery.....	70.15	

TABLE 11.—Results of concentration test with mill treating soft-ground ore.

Screen analysis of mill tailings, including overflow.					Zinc (Zn) content.		
Mesh.	Size of opening.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
3	6.680	0.263	<i>Per cent.</i> 16.23	<i>Per cent.</i>	<i>Per cent.</i> 0.48	4.9	
6	3.327	.131	30.20	46.43	.73	15.5	20.4
10	1.651	.065	17.50	63.98	.89	10.95	31.35
20	.833	.0328	11.03	74.96	1.34	10.4	41.75
25	.417	.0164	6.81	81.77	1.82	8.7	50.45
35	.208	.0082	1.86	83.63	2.90	3.8	54.25
100	.147	.0055	1.22	84.85	3.07	2.05	56.90
150	.104	.0041	.88	85.73	3.59	2.2	59.10
200	.074	.0029	.68	86.41	3.70	1.75	60.85
200	.074	.0029	13.59	100.00	4.10	39.15	100.00
Total or average.....			100.00		1.42	100.00	
Check sample.....					1.55		

Ore treated.....		Tons.	642
Concentrates (zinc) produced.....			40.32
Tailings.....			601.68
Zinc (Zn) in—		Per cent.	
Concentrates.....	58.60	23.627	
Tailings.....	1.55	9.3%	
Total zinc (metallic) in ore treated.....			32.953
Percentage of zinc recovery	71.70		

IMPORTANCE OF MILL TESTS.

When an ore is to be mined and treated, it is not always apparent, even to the most experienced metallurgist or millman, what method of procedure will give the highest recovery of the valuable minerals. After a concentrating plant or mill has been constructed and put in operation the mill superintendent or millman is apt to find that certain changes in construction are necessary to effect the best saving. However, tests should be made to determine what kind of work the mill, or a certain part of the concentration equipment, is doing, both before and after any changes are made. The millman in charge, however, should not discontinue the practice of making tests when he believes the most profitable recovery with the ore in question is being obtained, but should continue to make tests at frequent intervals throughout the life of the plant, because of the variations in the richness and character of the ore and in the quantity treated. The millman is obliged to consider what degree of recovery will yield the greatest profit; therefore the best technical results do not correspond to the best commercial results. Frequent testing,

however, will indicate where improvements can be made for increasing the saving of mineral; also the point of highest profits relative to the highest degree of efficiency practicable is brought out more definitely.

The concentration methods used in the Joplin district may seem crude to outsiders, but local practices adapted to local conditions should not be condemned without careful consideration. The penalties the smelters impose upon impure products and the conditions governing local prices must be taken into account. From these conditions and the usually low mineral content of the ore mined, it is obvious that the operator is guided more by the relative market values of his product than by the increase in saving that he might be able to effect through more elaborate methods or equipment.

VALUE OF SCREEN ANALYSES.

Except at a few of the mills in the district, little attempt is made at systematic mill tests to determine the percentage of recovery made in the mill or in the different steps of the milling process. This in spite of the fact that sampling tests should be made at every mill to aid the millman in detecting and rectifying unnecessary losses.

Screen analyses of the tailings should be made to determine in what sizes the mineral losses are greatest, so that a more definite idea can be obtained as to where further saving is possible. It may be found that the material coarser than 20-mesh does not contain mineral that would repay recovery, whereas that passing through a 20-mesh screen does. Consequently minor changes could be made to collect the material finer than 20-mesh for further treatment, or to improve that part of the mill that is losing the greater proportion of the ore particles finer than 20-mesh. If such screen analyses are made and each screen product is assayed, the results will indicate the metal content of the different sizes, and from the weights the percentage of loss in these sizes can be calculated.

In any screen analysis, the millman wishes to know what values are contained in the oversize or undersize of a screen used in the mill and having a certain size of screen openings. The selection of a set of screens that would be suited to every mill and every purpose would be difficult, if not impossible, as the perforated screens used are of various size openings, according to the local practice or the practice at any one mill. For making a screen test the millman can use whatever size of screen openings that, in his judgment, is best suited to the test he wishes to make. However, if he should wish to make an analysis of several sizes, especially of those intermediate of the sizes of screens used in the mill, the different size openings of the screens used should have a definite relation to each other. Many different makes of screens have the same mesh, but

owing to the different diameters of the wire used by the different manufacturers, the openings are not the same. The mesh, which is the number of openings per linear inch, means nothing if the size or diameter of the opening is not given.

Also, screen analyses should be made to determine the crushing efficiency of rolls and the screening efficiency of screens.

For all screen tests made, as given herein, a set of Tyler standard-screen-scale sieves was used, the openings of which are as follows:

TABLE 12.—*Sizes of Tyler standard-screen-scale sieves.*

Size of openings.		Mesh.	Diameter of wire.
Inches (ratio, $\sqrt{2}$ or 1.414).	Milli- meters.		
1.060	26.67		Inch. 0.149
.742	18.85		.135
.525	13.33		.106
.371	9.423		.092
.268	6.690	3	.070
.185	4.699	4	.065
.131	3.327	6	.036
.093	2.362	8	.032
.065	1.651	10	.025
.046	1.168	14	.025
.0328	.833	20	.0172
.0232	.589	28	.0125
.0164	.417	35	.0122
.0116	.295	48	.0092
.0082	.208	65	.0072
.0058	.147	100	.0042
.0041	.104	150	.0026
.0029	.074	200	.0021

COARSE CONCENTRATION.

SORTING.

When the ore has been hoisted from the shaft it is delivered on to a chute leading to a grizzly. This grizzly consists of iron bars, usually heavy rails, spaced 4 to 5 inches apart as shown in Plate XIII, A. The mineralized lumps in the oversize caught on the grizzly are broken with sledge hammers into pieces small enough to pass through; the boulders that are practically barren are sorted by hand and trammed to the waste-rock pile. The proportion of waste rock thus sorted out varies from a fraction of 1 per cent to 15 per cent, according to the character of the ore. The sorting is done by screenmen, better known locally as "cull men."

Sorting other than that by the screenmen is rarely practiced at the mines of the Joplin district. It is believed that sorting out the greater proportion of the large boulders of waste rock from the ore is of much importance if properly done. The reasons for this belief are perfectly clear. Eliminating a certain amount of waste rock increases the richness of the material retained for further treatment



A. METHOD OF DUMPING TUB ON GRIZZLY ABOVE HOPPER.



B. BLAKE CRUSHER, MOUNTED ABOVE PRIMARY ROLL

and also the relative capacity of the mill; makes possible a higher recovery; lessens the wear and tear on rolls, elevators, screens, and other equipment, and saves the power that would be consumed in treating waste rock. At one of the sheet-ground mines hand-picking the ore just before it entered the crusher was tried to good advantage, and a considerable saving in both mineral and costs per ton was said to have been made. If a system of sorting underground, especially in the sheet-ground mines, could be developed, or even a large proportion of waste rock sorted out by the screenmen at the grizzly, the writer believes that an appreciable saving would be effected.

CRUSHING AND SCREENING.

The undersize material, 5 inches or less in diameter, passes through the openings of the grizzly into the mill hopper below. From the hopper the ore is fed by means of an incline chute into a jaw crusher of the Blake type (Pl. XIII, *B*), where the larger size material is reduced to 2 inches or less in diameter, depending on the setting of the jaws of the crusher at the discharge end.

USE OF SHAKING SCREENS AHEAD OF CRUSHER.

At a few mills it was found that instead of an incline a perforated shaking screen having about $\frac{3}{8}$ to $\frac{1}{2}$ inch holes was used, a stream of water being played upon the material as it passed over the screen. Much of the finer material was washed through, thus eliminating unnecessary crushing of fines and increasing to some extent the relative capacity of the crusher and the rolls, the undersize being fed directly to the feed or "dirt" elevator. This feature is a useful one and has been effectively applied in some of the larger mills in other mining districts.

In the few mills in this district where this device has been tested there has been some difficulty in getting the undersize through the openings of the screen, chiefly because the narrow width of the screen causes the ore to bed too heavily and because a good shaking motion is lacking. In those districts where shaking screens are used at the head of the stamp or the crusher to by-pass a certain undersize, the screen is 4 to 5 feet wide at the feed end and tapers to 1 to 1½ feet in width at the discharge end. The shaking motion is effected by an eccentric. In the mills of the Joplin district, where such screens are used, the writer believes that the screening efficiency could be increased by using wider screens, thus permitting the ore to spread into a thinner bed, and by effecting a better shaking motion. A simpler device that might be used to by-pass the undersize into the feed elevator is an inclined grizzly with the bars set about three-fourths of an inch apart.

SCREEN TEST OF PRODUCT FROM SHAKING SCREEN.

To give some idea of the product from a shaking screen with $\frac{1}{2}$ -inch perforations, as used at the head of the crusher in one of the mills, the following screen analysis is given:

TABLE 13.—Results of screen analysis of undersize from shaking screen used ahead of crusher.

Size of screen openings (mesh).	Weight of screen product.	Cumulative weight.	Size of screen openings (mesh).	Weight of screen product.	Cumulative weight.
	<i>Per cent.</i>	<i>Per cent.</i>		<i>Per cent.</i>	<i>Per cent.</i>
≈ 0.525	4.96	-----	100	.52	98.32
8	48.21	48.17	150	.52	98.84
6	27.25	75.42	200	.62	99.46
10	11.18	86.60	<200	.54	100.00
20	5.37	91.97			
35	3.79	95.76	Total.....	100.00	-----
65	2.04	97.80			

$\approx$ Inch.

The table shows that 75 per cent of the undersize product from the shaking screen was oversize on the 6-mesh screen, and that only a small percentage of material larger than 0.525 inch, or approximately $\frac{1}{2}$ inch, passed through the $\frac{1}{2}$ -inch round openings in the shaking screen.

CRUSHERS, ROLLS, AND TROMMEL CIRCUIT.

As stated previously, the crushers used in the mills of this district are of the Blake type, and vary in size from 8 by 14 inches to 12 by 24 inches. More than one crusher is seldom used with the exception that if the mill is divided into two sections or units, as in a few of the larger plants, two crushers are employed. The usual running speed of crushers is 300 to 400 revolutions per minute.

The discharge from the crusher is fed directly to a set of rolls (see Pl. XIII, B) and is largely reduced to $\frac{1}{2}$ -inch size or less. As a rule only one set of rolls, better known as "first" rolls or primary rolls, is used for crushing to this size, but a few of the larger mills use two sets. Primary rolls generally measure 36 by 14 inches, although rolls measuring 30 by 14 and 42 by 14 inches are often used.

The roll shells are cast locally and consist of ordinary hard iron. Some of the mills are now using steel shells for recrushing middlings with satisfactory results. Closed rolls (faces touching) are generally used, although a few mills are equipped with spaced rolls which are run at a higher speed than the closed type. The advantage of spaced rolls is the production of less slimes.

The crushed ore from the primary rolls is elevated to a trommel, usually with $\frac{1}{2}$ -inch round-hole perforations; the undersize from the trommel goes to the roughing jigs and the oversize returns to one

or two secondary rolls, or return rolls, for further crushing. The product from the secondary rolls is fed back to the feed or "dirt" elevator and is returned to the trommel again. Thus the recrushing equipment forms a closed circuit so that all the material from the primary rolls must pass through the $\frac{1}{2}$ -inch openings of the trommel.

SCREEN TESTS OF PRODUCTS.

Some screen tests of the products from crushers, primary rolls, and return rolls are shown in Table 14 following.

TABLE 14.—Results of screen tests of feed and discharge of primary rolls in mill treating "sheet-ground" ore.

FIRST TEST.

Size of screen openings (mesh).	Feed to primary rolls (crusher discharge).		Discharge from primary rolls.	
	Weight of screen products.	Cumulative weight.	Weight of screen products.	Cumulative weight.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
•0.5	73.83	-----	32.78	-----
3	8.78	82.61	20.06	62.86
6	8.08	90.69	17.90	80.76
10	4.48	95.17	8.65	89.41
20	1.97	97.14	4.12	93.53
35	1.15	98.29	2.86	96.39
65	.68	98.97	1.77	98.16
100	.28	99.23	.56	98.72
150	.23	99.46	.47	99.19
200	.10	99.56	.21	99.40
<200	.44	100.00	.60	100.00
	100.00		100.00	

SECOND TEST.

•0.5	78.69	-----	38.26	-----
3	6.13	84.82	27.13	65.39
6	6.55	91.37	13.44	78.83
10	3.71	95.08	6.93	85.76
20	1.79	96.87	4.42	90.18
35	1.19	98.06	4.35	94.73
65	.75	98.81	2.79	97.52
100	.31	99.12	.86	98.88
150	.27	99.39	.62	99.00
200	.15	99.54	.28	99.28
<200	.46	100.00	.70	99.98
	100.00		99.98	

• Inch.

The table shows that more than 70 per cent of the material discharged from the crusher would remain on a $\frac{1}{2}$ -inch mesh screen, and that by passing this material through the first rolls the proportion of material coarser than $\frac{1}{2}$ inch is reduced about one half. The other screen products less than $\frac{1}{2}$ inch in size indicate how that part of the crusher discharge larger than $\frac{1}{2}$ -inch size, is distributed through the different sizes below $\frac{1}{2}$ inch.

The oversize from the trommels handling the material from the feed or "dirt" elevator is crushed by one or two sets of return rolls, usually 30-inch. As previously mentioned, the crushing is in closed circuit, all the material being crushed to pass the openings of the trommel before going to the jigs.

The results of screen analyses of the feeds to return rolls and their products are given in Table 15, following.

TABLE 15.—*Results of screen analyses of feed and product of return rolls in mill treating "hard-ground" ore.*

Screen.			First series of tests.				Second series of tests.			
			Feed to return rolls.		Product from return rolls.		Feed to return rolls.		Product from return rolls.	
Size of mesh.	Size of opening.		Weight of screen products.	Cumulative weight	Weight of screen products.	Cumulative weight.	Weight of screen products.	Cumulative weight.	Weight of screen products.	Cumulative weight.
	Mm.	Inches.								
			<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
3	6.680	0.525	31.39		3.00		30.82		2.15	
6	3.327	.263	64.23	95.62	43.53	46.53	59.23	20.05	44.06	46.21
10	1.651	.131	3.75	99.37	26.70	73.23	8.03	98.08	29.56	75.77
20	.833	.065	.27	99.64	11.94	85.17	.79	98.87	10.59	86.36
35	.417	.0164	.08	99.72	5.94	91.11	.36	99.23	5.38	91.74
65	.208	.0082	.08	99.80	3.55	94.66	.28	99.51	3.49	95.23
100	.147	.0058	.20	100.00	1.93	96.59	.20	99.71	1.81	97.04
150	.104	.0041			.88	97.47	.11	99.82	.81	97.85
200	.074	.0029			.19	97.66	.04	99.86	.13	97.98
<200	.074	.0029			.65	98.31	.04	99.90	.70	98.68
					1.69	100.00	.10	100.00	1.32	100.00
Total.....			100.00		100.00		100.00		100.00	

In the two examples of screen tests given above, the rolls were doing good work, better than the average done in the mills throughout the district. However, the relatively more efficient crushing is partly the result of efficient screening by the screen trommels.

CRUSHING OF "CHATS."

The next step in crushing for coarse concentration is to crush the middlings from the different jigs. This material, which needs further grinding and concentration, is locally known as "chats," the set of rolls crushing it is known as "chat" rolls, and the jigs that treat the crushed product separately are called "chat" jigs.

It is in crushing chats that the least effective crushing is done in the mills of the district. As an example of rather poor results in crushing chats, screen analyses of the feed and the product from a set of 24-inch rolls are shown in Table 16 herewith.

TABLE 16.—Results of screen analyses of feed and product of "chat" rolls in mill treating "hard-ground" ore.

Sieve.			Feed to chat rolls.		Product from chat rolls.	
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	Weight of screen products.	Cumulative weight.
	Mm.	Inches.				
			<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
3	6.680	0.263	18.16	-----	10.48	-----
6	3.327	.131	69.02	85.18	68.98	79.46
10	1.651	.065	11.96	97.14	16.10	95.56
20	.833	.0328	1.88	99.00	1.91	97.47
35	.417	.0164	.78	99.78	1.49	98.96
65	.208	.0082	.11	99.89	.44	99.40
100	.147	.0068	.04	99.93	.15	99.55
150	.104	.0041	.01	99.94	.07	99.62
200	.074	.0029	.02	99.96	.07	99.69
<200	.074	.0029	.04	100.00	.31	100.00
Total.....			100.00	-----	100.00	-----

Uneven wear of the shells, of the usual cast-iron type used so extensively throughout the district, is responsible for the seemingly poor work of these rolls. For material of this size steel shells are much more efficient and should be used. Returning so much oversize to the trommel reduces its screening efficiency.

COARSE SCREENING.

Cylindrical trommels are the screens most commonly used throughout the district for coarse work. Both trommel and shaking screens are used for fine screening although the former finds more favor among the mill men.

The screening efficiency of a trommel depends on the size of the trommel, both its diameter and length, its speed, its slope, the size of screen opening, and the quantity and rate of feed. In the mills of the Joplin district the trommels are run at speeds ranging from 15 to 28 revolutions per minute, and have slopes of five-eighths of an inch to 2 inches per foot. For coarse screening the trommels usually are 36 by 96 inches, 48 by 96 inches, and 48 by 144 inches, with three-eighths to five-eighths inch openings at the head of "rougher" jigs and one-fourth to one-eighth inch openings at the head of sand or "chat" jigs.

It has been stated that the discharge from the feed or "dirt" elevator is fed to a trommel; the undersize from the trommel passes to the rougher jigs, while the oversize is crushed through the "return" rolls and returned to the feed elevator. Where the entire feed passes eventually through the openings of the trommel, the trommel being in closed circuit with the return rolls, the millman is likely to be less careful in determining the best conditions for most efficient and economical screening and capacity, as long as the main purpose of

the screen is accomplished. The writer has frequently found trommels running too fast for efficient screening. Increasing the speed beyond certain limits, lessens the screening efficiency, increases the quantity of oversize, adds to the wear on the rolls from grinding additional oversize, and the undersize, which should have passed through the screen openings, is reduced still finer by the rolls in closed circuit.

In the Joplin district the ore is screened wet. If the trommel has not sufficient slope with reference to the speed and the diameter, the feed is liable to be too heavy and prevent the undersize from working freely through the screen. This is especially true of the finer sized material when not enough water is used. The mill man should determine as nearly as possible the slope and speed best suited to the trommel for the given quantity of feed and size of material.

EFFICIENCY OF TROMMELS.

To show the screening efficiency of trommels in some of the Joplin mills the results of a few tests are given below.

TEST 1.—Screening efficiency of trommel with $\frac{1}{2}$ -inch openings.

[Trommel size, 48 by 144 inches; speed, 25 r. p. m.; slope, $1\frac{1}{2}$ inches per foot.]

$$\text{Efficiency} = \frac{\text{Weight of undersize passing through the screen.}}{\text{Total weight of undersize in the feed.}}$$

Product.	Trommel feed.	Trommel oversize.
Oversize on $\frac{1}{2}$ -inch screen, per cent.....	17.2	44.7
Undersize through $\frac{1}{2}$ -inch screen, per cent.....	82.8	55.3
	100.0	100.0

$$\frac{17.2 \times 55.3}{44.7} = 21.28; \text{ efficiency} = \frac{(82.8 - 21.28) \times 100}{82.8} = 74.06 \text{ per cent.}$$

TEST 2.—Screening efficiency of trommel with $\frac{3}{8}$ -inch openings.

[Trommel size, 48 by 144 inches; speed, 25 r. p. m.; slope, $1\frac{1}{2}$ inches per foot.]

Product.	Trommel feed.	Trommel oversize.
Oversize on $\frac{3}{8}$ -inch screen, per cent.....	21.4	42.1
Undersize on $\frac{3}{8}$ -inch screen, per cent.....	78.6	57.9
	100.0	100.0

$$\frac{21.4 \times 57.9}{42.1} = 29.43; \text{ efficiency} = \frac{(78.6 - 29.43) \times 100}{78.6} = 62.55 \text{ per cent.}$$

TEST 3.—Screening efficiency of trommel with $\frac{1}{8}$ -inch openings.

[Trommel size, 48 by 144 inches; speed, 21 r. p. m.]

Product.	Trommel feed.	Trommel oversize.
Oversize on $\frac{1}{8}$ -inch screen, per cent.....	23.1	64.5
Undersize through $\frac{1}{8}$ -inch screen, per cent.....	76.9	35.5
	100.0	100.0

$$\frac{23.1 \times 35.5}{64.5} = 12.71; \text{ efficiency} = \frac{(76.9 - 12.71) \times 100}{76.9} = 83.47 \text{ per cent.}$$

TEST 4.—*Screening efficiency of trommel with ½-inch openings.*

[Trommel size, 48 by 144 inches; speed, 21 r. p. m.; slope, 1½ inches per foot.]

Product.	Trommel feed.	Trommel oversize.
Oversize on ½-inch screen, per cent.....	13.3	40.3
Undersize through ½-inch screen, per cent.....	86.7	59.7
	100.0	100.0

$$\frac{13.3 \times 59.7}{40.3} = 19.7; \text{ efficiency} = \frac{(86.7 - 19.7) \times 100}{86.7} = 77.28 \text{ per cent.}$$

TEST 5.—*Screening efficiency of trommel with ¼-inch openings.*

[Trommel size, 48 by 96 inches; speed, 26 r. p. m.; slope, 1½ inches per foot.]

Product.	Trommel feed.	Trommel oversize.
Oversize on ¼-inch screen, per cent.....	15.3	29.7
Undersize through ¼-inch screen, per cent.....	84.7	70.3
	100.0	100.0

$$\frac{15.3 \times 70.3}{29.7} = 36.22; \text{ efficiency} = \frac{(84.7 - 36.22) \times 100}{84.7} = 57.24 \text{ per cent.}$$

In most mills the poor screening efficiency of trommels results from too high speed. Usually the best speed for a trommel, under proper conditions, is 15 to 21 revolutions per minute, and the trommel should not, as a rule, be run much faster or slower than these limits.

JIGGING.

The Cooley jig (Pl. XIV and fig. 9), similar in principle to the Harz jig, is the most common type of jig used in this district. It is of the fixed-sieve type, the water being forced up and down through the stationary screens or grates by the action of plungers in adjacent compartments. The number and the size of the compartments for each jig depend on the size and character of the ore treated.

OUTLINE OF CONCENTRATION SYSTEM.

In general, a system of "roughing" and "cleaning" is followed. The feed is given a preliminary cleaning on one or two "rougher" jigs that eliminates the greater proportion of waste material. Then the enriched product, which will assay 10 to 25 per cent zinc, is treated on a "cleaner" jig, bringing the zinc tenor up to 50 to 60 per cent. The "chats" or middlings from the rougher and the cleaner jigs, which together will assay 4 to 8 per cent zinc, are recrushed and either returned to the rougher-jig feed or treated separately over a "chat" jig. The tailings from the rougher jig are usually dewatered by means of a dewatering trommel with 1½-mm. to 2-mm. openings, over the outside of which the tailings pass as the trommel slowly revolves. The undersize from this trommel flows to the settling tanks and the oversize to the tailings elevator as waste. The overflow from the

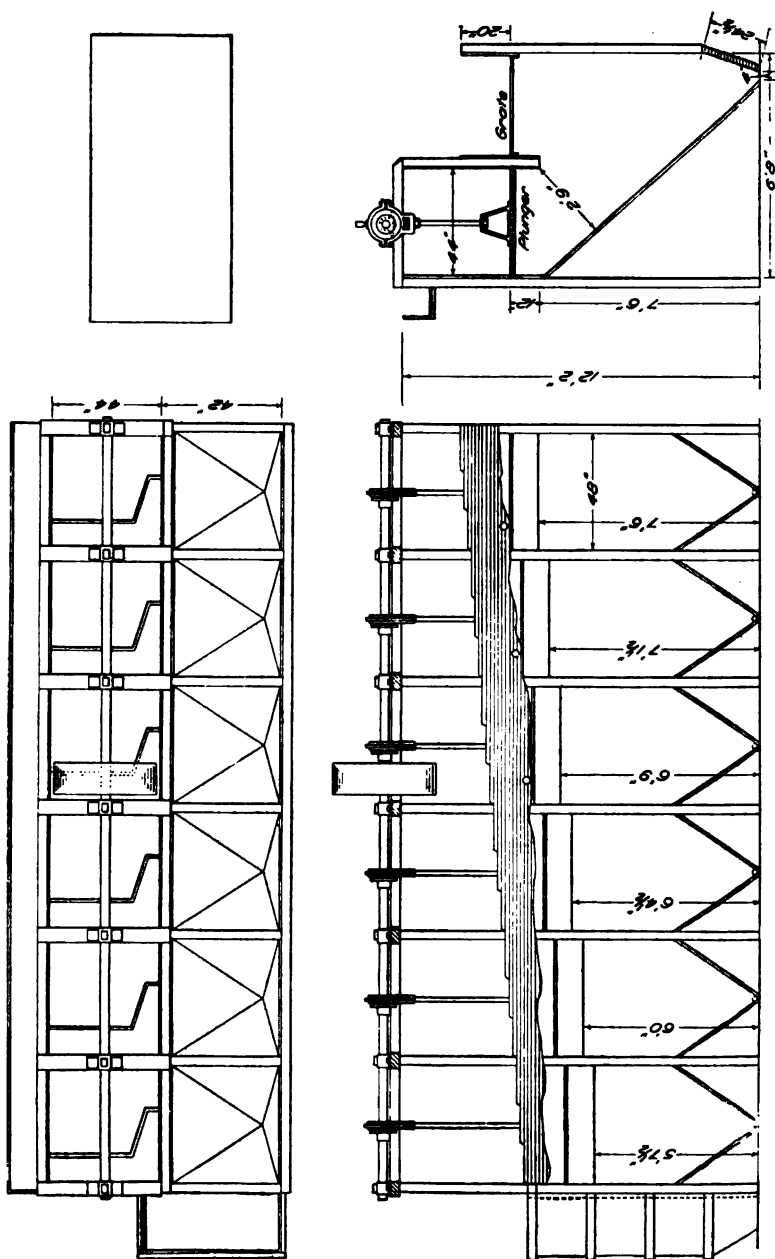
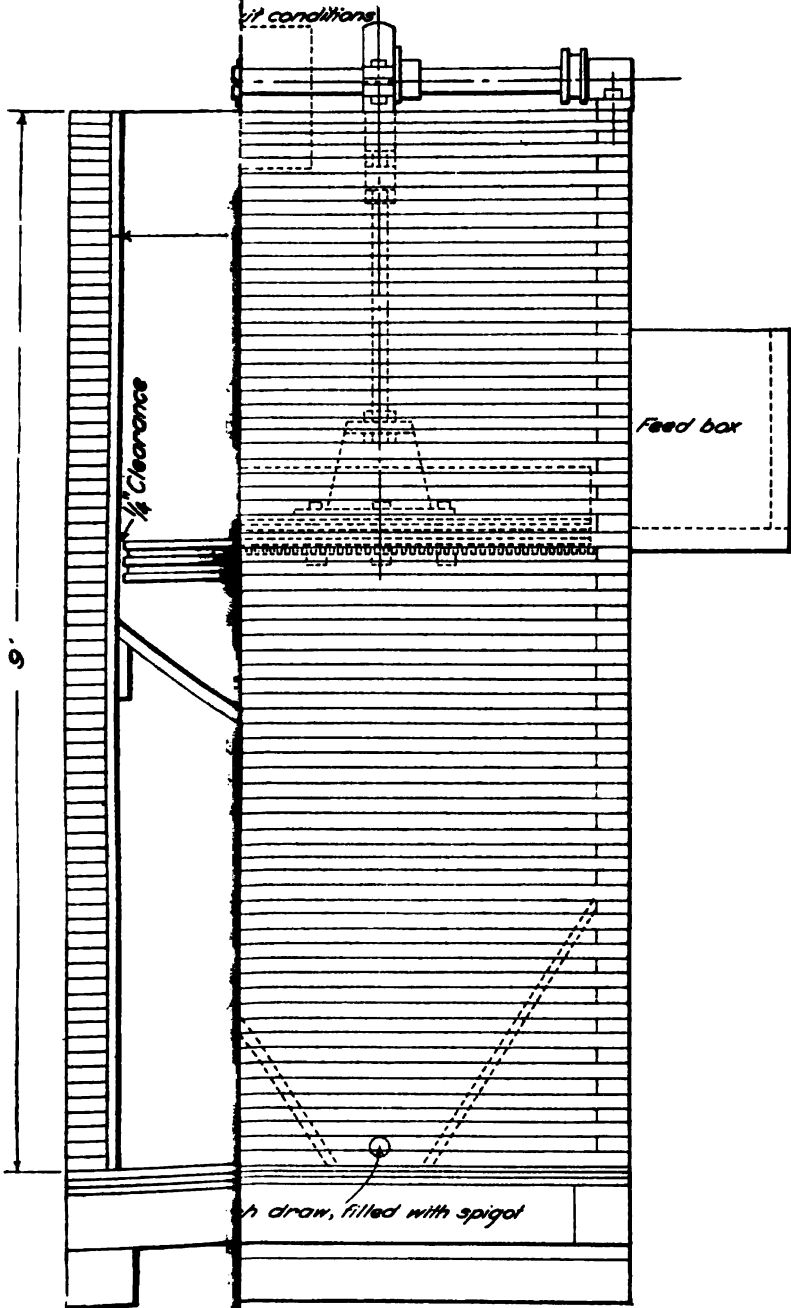


FIGURE 9.—A recent design of Cowley jig.



tailing end of the cleaner jig, and other jigs if used, also passes to the settling tanks for subsequent treatment.

The rougher jigs usually have 5 or 6 cells with a screening or grate area ranging from 30 by 42 inches to 36 by 48 inches. The speed of the shaft connecting the plungers and the eccentrics varies from 90 to 120 revolutions per minute; the length of stroke of the plungers ranges from $\frac{3}{4}$ inch to $1\frac{1}{4}$ inches. The cleaner jigs each have 6 or 7 cells with a grate area somewhat smaller than that of the rougher jigs. The speed of the shaft connecting the plungers and the eccentrics of cleaner jigs is 160 to 200 revolutions per minute; the length of stroke ranges from $\frac{3}{4}$ to $\frac{7}{8}$ inch. The chat or sand jigs, which are not commonly used, are smaller in size and usually have 4 or 5 cells each, and are operated at a higher speed and shorter stroke.

The material fed to the first cell of the rougher jig is, as a rule, not graded or classified. A bed 5 to 6 inches deep is formed as a result of the pulsating action of the water, the lighter gangue material, such as flint, rises to the top of the bed, the heavier free grains of lead and zinc minerals working down to the bottom. The downward suction stroke draws the finer grains of mineral through the screen or grate openings into the hutch of each compartment. The strength of the suction is increased by leaving the gates of the hutch partly open. The material that accumulates in the hutch is known as "smittem," and is further treated on the cleaner jig. The bed products of the first two or three cells are also drawn off and pass with the hutch product to the cleaner jig. The chats or particles containing included lead and zinc mineral, usually from the last hutch and the last two or three beds, are drawn off and reground before further treatment.

Various tests conducted by the writer at different mills throughout the district indicate that a few minor changes in the flow sheet or treatment of the ore would at many of the mills effect more efficient jigging and a greater saving. To publish the data on each test would require too much space, but the results of some of the tests are given to bring out the important points under discussion.

EFFICIENCY OF JIGS.

A few results of efficiency tests made on rougher jigs are given to show the average percentage of efficiency of rougher jigs on zinc-blende ore.

The treatment of rougher jigs can be considered a closed circuit in each example given, because the chats or middlings were being reground and returned to the feed of the rougher jig. On this basis, the percentage of efficiency or zinc recovery can readily be calculated by the formula

$$R = \frac{100 \times c(h-t)}{h(c-t)}$$

where R represents percentage of efficiency, h represents the zinc content of the head or feed to the rougher jig, c the zinc content of the concentrate product from the jig, and t the zinc content of the tailing as it leaves the jig. In the first example the feed (h) assayed 2.85 per cent zinc, the concentrate (c) 25.5 per cent zinc, and the tailing (t) 1.10 per cent zinc. Substituting these figures in the formula and solving show that the efficiency or percentage of zinc recovery from the rougher jig was 64.17 per cent. In other words this jig saved only 64.17 per cent of the total zinc content in the feed. However, in this mill the tailing, discharged from the last cell, was dewatered over a trommel before leaving the mill as waste. Dewatering eliminated a considerable proportion of slime from the tailing, and as the slime, as a rule, assays considerably higher than the coarser material, the tailing assay was lowered from 1.10 to 0.70 per cent zinc.

RESULTS OF EFFICIENCY TESTS OF ROUGHER JIGS.

EXAMPLE 1. SHEET-GROUND ORE.

	Zinc content, per cent.
Feed or head (h)	2.85
Concentrate (c)	25.5
Tailing (t) discharge from last cell	1.10
Dewatered tailing	.70

$$R = \frac{100 \times c (h - t)}{h (c - t)} = \frac{100 \times 25.5 (2.85 - 1.10)}{2.85 (25.5 - 1.80)} = 64.17 \text{ per cent.}$$

EXAMPLE 2. SHEET-GROUND ORE.

Feed or head (h)	1.13
Concentrate (c)	15.0
Tailing (t) discharge from last cell	.50
Dewatered tailing	.38

$$R = \frac{100 \times c (h - t)}{h (c - t)} = \frac{100 \times 15.0 (1.13 - 0.50)}{1.13 (15.0 - 0.50)} = 57.67 \text{ per cent.}$$

EXAMPLE 3. HARD-GROUND ORE OTHER THAN SHEET-GROUND.

Feed or head (h)	1.20
Concentrate (c)	9.57
Tailing (t) discharge from last cell	.55
Dewatered tailing	.34

$$R = \frac{100 \times c (h - t)}{h (c - t)} = \frac{100 \times 9.57 (1.20 - 0.55)}{1.20 (9.57 - 0.55)} = 57.47 \text{ per cent.}$$

EXAMPLE 4. ORE IN MIAMI DISTRICT.

Feed or head (h)	1.99
Concentrate (c)	14.1
Tailing (t) discharge from last cell	.87
Dewatered tailing	.82

$$R = \frac{100 \times c (h - t)}{h (c - t)} = \frac{100 \times 14.1 (1.99 - 0.87)}{1.99 (14.1 - 0.87)} = 59.98 \text{ per cent.}$$

From the results given, and others on hand, it can be stated that 20 to 30 per cent of the total zinc in the ore treated is lost in the first stage of concentration over rougher jigs. Hence, some tests are presented to show what the chief causes of this loss are and how it can be lessened.

LOSSES FROM JIGS.

The feed to the rougher jig, which in most mills is ungraded material, consists of both coarse and fine material. The results of screen analyses of the feed, the total tailing from the rougher jig, and the dewatered tailing, and the assays of the various screen products of two mills are given in Tables 17 and 18 following. Table 17 represents the results of treating sheet-ground ore over a rougher jig. Table 18 represents the results of treating of a chatty ore in the Miami district over a rougher jig.

TABLE 17.—Results of screen analyses in treatment of sheet-ground ore over rougher jig.

A. FEED TO THE ROUGHER JIG.

Screen.			Weight of screen products.	Cumulative weight.	Zinc content by assay.	Percentage of total zinc.	Cumulative zinc content.
Size of mesh.	Size of opening.						
	Mm.	Inches.					
			<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>		<i>Per cent.</i>
3	6.680	0.263	9.26	9.26	0.55	4.53	4.53
6	3.327	.131	44.08	53.34	.65	26.18	30.71
10	1.651	.065	22.10	75.44	1.02	20.07	50.07
20	.833	.0328	10.14	85.58	1.16	10.46	61.24
35	.417	.0164	5.55	91.13	1.68	8.30	69.54
65	.206	.0082	3.16	94.29	2.15	6.04	75.58
100	.147	.0058	1.27	95.56	3.17	3.60	79.18
150	.104	.0041	.97	96.53	3.74	3.22	82.40
200	.074	.0029	.19	96.72	4.52	.79	83.19
<200	.074	.0029	3.28	100.00	5.77	16.81	100.00
Total or average.....			100.00	-----	1.13	100.00	-----

B. TAILING FROM ROUGHER JIG BEFORE PASSING DEWATERING SCREEN.

3	6.680	0.263	7.66	7.66	0.53	8.06	8.06
6	3.327	.131	46.80	54.46	.35	32.48	40.54
10	1.651	.065	23.83	78.29	.32	15.10	55.64
20	.833	.0328	9.57	87.86	.26	4.98	60.57
35	.417	.0164	4.25	92.11	.23	1.94	62.51
65	.206	.0082	2.08	94.19	.32	1.31	63.82
100	.147	.0058	.98	95.17	.51	.97	64.79
150	.104	.0041	1.06	96.23	1.14	2.40	67.19
200	.074	.0029	.15	96.38	1.85	.55	67.74
<200	.074	.0029	3.62	100.00	4.50	32.26	100.00
Total or average.....			100.00	-----	.50	100.00	-----

C. DEWATERED TAILING FROM ROUGHER JIG.

3	6.680	0.263	7.20	7.20	0.51	9.69	9.69
6	3.327	.131	50.49	57.69	.36	42.30	51.99
10	1.651	.064	27.81	85.50	.34	25.04	77.03
20	.833	.0328	8.72	94.22	.28	6.46	83.49
35	.417	.0164	2.84	97.06	.25	1.91	85.40
65	.206	.0082	.98	98.04	.27	.69	86.09
100	.147	.0058	.41	98.45	.43	.46	86.55
150	.104	.0041	.39	98.84	.96	.98	87.53
200	.074	.0029	.07	98.91	1.90	.35	87.88
<200	.074	.0029	1.09	100.00	4.20	12.12	100.00
Total or average.....			100.00	-----	.38	100.00	-----

TABLE 18.—Results of screen analyses in treatment of chatty ore over rougher jig.

(Miami district.)

A. FEED TO THE ROUGHER JIG.

Screen.			Weight of screen products.	Cumulative weight.	Zinc content by assay.	Percent- age of total zinc.	Cumula- tive zinc content.
Size of mesh.	Size of opening.						
	Mm.	Inches.					
			<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>		<i>Per cent.</i>
3	6.680	0.263	2.00	2.00	1.20	1.21	1.21
6	3.327	.131	26.20	28.20	1.25	16.48	17.69
10	1.651	.065	21.26	49.46	1.32	14.13	31.82
20	.833	.0328	9.16	58.62	1.82	8.39	40.21
35	.417	.0164	6.32	64.94	3.20	10.17	50.35
65	.208	.0082	6.78	71.72	4.00	13.66	64.04
100	.147	.0058	5.15	76.87	2.90	7.52	71.56
150	.104	.0041	6.05	82.92	2.65	8.07	79.63
200	.074	.0029	4.65	87.57	2.67	6.24	85.87
<200	.074	.0029	12.43	100.00	2.25	14.13	100.00
Total or average.....			100.00		1.99	100.00	

B. TAILING FROM ROUGHER JIG BEFORE PASSING DEWATERING SCREEN.

3	6.680	0.263	1.91	1.91	0.75	1.65	1.65
6	3.327	.131	48.43	50.34	.75	41.76	43.41
10	1.651	.065	21.87	72.21	.85	21.43	64.84
20	.833	.0328	5.92	78.13	.90	6.15	70.99
35	.417	.0164	2.67	80.80	.90	2.75	73.74
65	.208	.0082	2.20	83.00	.85	2.20	75.94
100	.147	.0058	2.10	85.10	.75	1.76	77.70
150	.104	.0041	2.67	87.77	.80	2.42	80.12
200	.074	.0029	2.20	89.97	1.25	3.18	83.30
<200	.074	.0029	10.03	100.00	1.45	16.70	100.00
Total or average.....			100.00		.87	100.00	

C. DEWATERED TAILING FROM ROUGHER JIG.

3	6.680	0.263	1.14	1.14	0.75	1.05	1.05
6	3.327	.131	55.48	56.62	.75	50.93	51.98
10	1.651	.065	26.60	83.22	.85	27.70	79.68
20	.833	.0328	8.39	91.61	.85	8.71	88.39
35	.417	.0164	1.61	93.22	.95	1.86	90.25
65	.208	.0082	.90	94.12	.80	.87	91.12
100	.147	.0058	.81	94.93	.75	.75	91.87
150	.104	.0041	.81	95.74	.95	.83	92.30
200	.074	.0029	.47	96.21	1.20	.70	93.50
<200	.074	.0029	3.79	100.00	1.40	6.50	100.00
Total or average.....			100.00		.82	100.00	

Part A of Table 17 shows that 3.28 per cent of the feed (crushed sheet-ground ore) to the rougher jig consisted of material finer than 200-mesh; its zinc content was 5.77 per cent, which represented 16.81 per cent of the total zinc in the feed. Part B of Table 17, which represents the screen analysis of the tailing before it was dewatered, shows that the assay value of the material finer than 200-mesh was 4.50 per cent zinc. The percentage of material finer than 200-mesh in the tailing was 3.62, which represented 32.26 per cent of the total zinc in the tailing. The screen analysis of the dewatered tailing, part C of Table 17, showed that there was still 1.09 per cent of the material finer than 200-mesh contained in the tailing, with an assay value of 4.20 per cent of zinc, representing 12.12 per cent of the total zinc contained in the dewatered tailing that was discarded.

DESLIMING AT HEAD OF ROUGHER JIGS ADVISABLE.

It is reasonable to believe, from these figures alone, that desliming at the head of the rougher jig would be advisable. As the results in Table 17 show the necessity of desliming at the head of the rougher jig when a sheet-ground ore is treated, the results in Table 18 show a much greater proportion of slimes in the feed to the rougher when the ore is "chatty" or more or less disseminated and is ground finer.

However, the importance of desliming is more strongly emphasized by the fact that the dewatering screens as used at the end of the jigs in this district are, as a rule, not very efficient, so that fine mineral is lost in the jig tailing. To show this more fully, the screening efficiency of the dewatering screens used at the end of the two rougher jigs under discussion was determined. It should be stated that the main object of the dewatering screens in this district is not only to eliminate the water from the tailing, but also to collect as much as possible of the fines from the tailing overflow for further treatment on tables.

EXAMPLE 1.—EFFICIENCY OF DEWATERING SCREEN WITH 1.5-MM. OPENINGS, AT END OF ROUGHER JIG TREATING SHEET-GROUND ORE.

Product, size of opening.	Feed, per cent.	Oversize, per cent.
Over 1.5 mm.....	80.1	87.5
Through 1.5 mm.....	19.9	12.5
	100.0	100.0

$$\text{Weight of undersize in the oversize} = \frac{80.1 \times 12.5}{87.5} = 11.44.$$

$$\text{Screening efficiency} = \frac{\text{Weight of undersize passing through screen}}{\text{Total weight of undersize in feed to screen}} = \frac{(19.9 - 11.44) \times 100}{19.9} = 42.51 \text{ per cent.}$$

The efficiency of a dewatering screen with 1.5-mm. openings used at the end of a rougher jig treating a chatty ore of the Miami district showed that 52.47 per cent of the undersize in the feed to the screen was screened through the 1.5-mm. openings. In this mill the dewatering screen used was an ordinary flat shaking screen, instead of the usual revolving trommel. The screening efficiency was as follows:

EXAMPLE 2.—EFFICIENCY OF FLAT DEWATERING SCREEN IN MILL TREATING CHATTY ORE.

Product, size of opening.	Feed, per cent.	Oversize, per cent.
Over 1.5 mm.....	73.7	85.5
Through 1.5 mm.....	26.3	14.5
	100.0	100.0

$$\text{Weight of undersize in the oversize} = \frac{73.7 \times 14.5}{85.5} = 12.5.$$

$$\text{Screening efficiency} = \frac{\text{Weight of undersize passing through screen}}{\text{Total weight of undersize in feed to screen}} = \frac{(26.3 - 12.5) \times 100}{26.3} = 52.47 \text{ per cent.}$$

To show more fully the screening efficiencies of dewatering screens in various mills, two more examples are given.

EXAMPLE 3.—EFFICIENCY OF DEWATERING SCREEN IN MILL TREATING HARD-GROUND ORE OTHER THAN SHEET-GROUND.

[Feed to rougher jig crushed to $\frac{1}{8}$ inch.]

Product, size of opening.	Feed, per cent.	Oversize, per cent.
Over 1.5 mm.....	81.6	91.0
Through 1.5 mm.....	18.4	9.0
	100.0	100.0

$$\text{Weight of undersize in the oversize} = \frac{81.6 \times 9}{91.0} = 8.07.$$

$$\text{Screening efficiency} = \frac{\text{Weight of undersize passing through screen}}{\text{Total weight of undersize in feed to screen}} = \frac{(18.4 - 8.07) \times 100}{18.4} = 65.2 \text{ per cent.}$$

EXAMPLE 4.—EFFICIENCY OF DEWATERING SCREEN IN MILL TREATING SHEET-GROUND ORE.

[Feed to rougher jig crushed to $\frac{1}{2}$ inch.]

Product, size of opening.	Feed, per cent.	Oversize, per cent.
Over 2 mm.....	63.8	64.8
Through 2 mm.....	36.2	15.2
	100.0	100.0

$$\text{Weight of undersize in the oversize} = \frac{63.8 \times 15.2}{84.8} = 11.43.$$

$$\text{Screening efficiency} = \frac{\text{Weight of undersize passing through screen}}{\text{Total weight of undersize in feed to screen}} = \frac{(36.2 - 11.43) \times 100}{36.2} = 68.37 \text{ per cent.}$$

The average efficiency of dewatering screens in the mills of the Joplin district can be given as 40 to 60 per cent. The results in example 4 show the highest screening efficiency obtained from a number of dewatering screens tested. The screen consisted of a 5-foot trommel running at a speed of $1\frac{1}{2}$ revolutions per minute. This relatively higher efficiency was attained, however, only after careful adjustment of the feed. As the efficiency of the trommel or the quantity of undersize passing through the openings depends on the flow of water used with the tailing feed to force the undersize through, it is essential that the tailing flow strike the trommel in such a way that most of the water will pass through the openings. In a few mills it was noticed that the line of flow was more or less tangent to the surface of the revolving screen so that a large part of the water and fines had no chance to pass through the openings.

In some mills no dewatering screens are used, but a rectangular box is added to the end of the jig. From the bottom of this box the coarse tailing is discharged through a spigot while the fines and water are

allowed to overflow at the top. In a few mills water is discharged under pressure into the box to cause as much fines and slimes as possible to overflow with the water, thus eliminating them from the spigot discharge, but with rather unsatisfactory results on the whole.

Another point in favor of desliming at the head of the rougher jig is that the greater the content of slimes in the feed the more will the valuable or fine mineral particles be forced over the jig by the flow of top water, especially as the quantity of water added to any cell of the jig increases the top water of each succeeding cell. Also, better work can be done with the jig by having clear top water, which is another reason, when the material is not graded or sized, why the feed to the rougher should be deslimed. However, desliming the jig feed would not necessarily mean eliminating the dewatering screen at the end of the rougher jig.

SUGGESTED METHODS OF DESLIMING AHEAD OF ROUGHER JIG.

Several ways of desliming at the head of the rougher jig could be followed. A simple and cheap device would be a V-box with one or two discharge spigots attached to the bottom, the overflow of fines and slimes at the top being assisted by water under hydraulic pressure. A somewhat more efficient means would be the use of a trommel screen with $1\frac{1}{2}$ or 2 mm. openings, although the upkeep would be higher. The feed would discharge into one end of the trommel on the inside, the undersize working through the openings in the usual way. This method has proved satisfactory in the desliming and re-treatment of tailings.

Another effective method of desliming at the head of rougher jigs would be the use of a large size classifier similar to the Akins. The Akins is mentioned here as it is simple and would not require any extensive changes in the arrangement of the mill as far as headroom is concerned nor loss in drop from the "dirt" or feed trommel to the rougher jig. The cost of this desliming screen would be about \$1,000 installed.

The Woodbury desliming jig might be used to good advantage. This jig would deslime the feed and also eliminate a large proportion of the lead mineral, thus permitting the rougher jig to do more efficient work on the remaining zinc ore. Such a jig would increase the efficiency of the roughing system, but the first cost would be higher than for any of the other devices mentioned, and in most mills more headroom would be needed and the feed trommels and feed elevators would have to be rearranged.

DESLIMING AT SECOND OR THIRD CELL OF ROUGHER JIG.

Instead of desliming at the head of the rougher jig a means has been devised for desliming the material after it has reached the second or

third cell of the jig (see fig. 10). A wooden baffle is fastened to the ends of two strips of wood, the opposite ends of which are pivoted to the inner sides of a jig cell. This baffle is placed at right angle to the flow and adjusted to move up and down with the pulsating action of the plungers while resting on top of the bed, the water and slimes being removed just before they flow over the partition between the cells. The slimes and top water are diverted off the bed at the plunger side of the jig and by means of a launder conveyed to settling tanks; the gangue and remaining ore pass under the baffle into the next cell. Not only is a large proportion of slimes and excessive

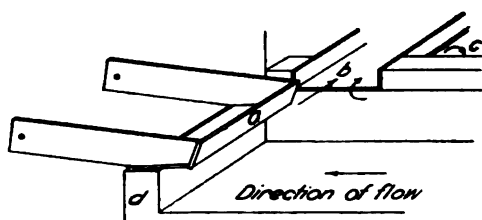


FIGURE 10.—Device used for desliming top water on jigs. a, Baffle; b, top water and slimes to settling tanks; c, plunger cells; d, partition between cells.

top water removed by having these baffles at one or more cells, but the speed of the surface currents or top water is reduced, thus providing a steadier discharge with less disturbance of the bed at the head of each succeeding cell. Such baffles, in the few mills where their use has been observed by

the writer, eliminate sufficient slimes from the top water to give practically clear water in the last cell of the jig.

RETREATING RECRUSHED CHATS OVER SEPARATE JIGS AND TABLES.

The chats from the rougher jig are recrushed and treated over a chat jig or returned to the rougher jig. If treated over a chat or sand jig, the crushed material is usually cleaned on the jig direct, without a preliminary roughing. It would seem advisable, where the quantity of chats produced from both the rougher and cleaner jigs is relatively large, to treat the recrushed chats over a rougher chat jig and the roughed product over a cleaner chat jig. If only a moderate quantity of chats were produced, they could be crushed to a certain size, about one-eighth to three-sixteenths inch, and then treated over a rougher chat jig. The enriched product obtained could be given a final treatment over the main cleaner jig, and thus a cleaner chat jig would not be needed. The middling or chats from the rougher chat jig could be either returned to the chat rolls and retreated over the chat jig or sent to sand rolls and treated on tables.

Another method of treating the chats from rougher and cleaner jigs would be to crush to table size, give the crushed material a preliminary treatment on roughing tables to eliminate much of the waste sand, and run the product over the usual sand tables.

One of the chief reasons for treating the chats separately, and not returning them to the rougher jig feed, is that the recrushed chats

is an enriched product, assaying 4 to 8 per cent zinc, whereas the feed to the rougher jig assays only about 1.5 to 3 per cent zinc. A higher recovery is possible with a richer feed than with a leaner one; hence a greater percentage of the zinc in the relatively richer chats could be saved by treating them separately than by returning them to the rougher jig feed and mixing them with the relatively much lower feed. Screen analyses and assays showing the differences in zinc content of chats from both rougher and cleaner jigs as compared to the feed to the rougher jigs are shown in Table 19 following.

TABLE 19.—Results of screen analyses of feeds and products of rougher and cleaner jigs.

FEED TO ROUGHER JIG NO. 1.

Screen.			Weight of screen products.	Cumulative weight.	Zinc content by assay.	Percentage of total zinc.	Cumulative zinc content.
Size of mesh.	Size of opening.						
	Mm.	Inches.					
			<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>		<i>Per cent.</i>
3	6.680	0.263	12.16	1.24	4.8		
6	3.327	.131	33.71	45.87	20.9		25.7
10	1.651	.065	17.65	63.52	15.85		41.55
20	.833	.0328	12.01	75.53	12.4		53.95
35	.417	.0164	10.55	86.08	13.4		67.35
65	.208	.0082	5.67	91.75	11.75		79.10
100	.147	.0058	2.25	94.00	10.15		86.30
150	.104	.0041	.60	94.60	10.00		88.20
200	.074	.0029	2.37	96.97	9.10		94.30
<200	.074	.0029	3.03	100.00	5.93		100.00
Total or average.....			100.00		3.16	100.00	

FEED TO ROUGHER JIG NO. 2.

3	6.680	0.263	15.73	1.08	6.8		
6	3.327	.131	34.50	50.23	24.4		31.2
10	1.651	.065	19.44	69.67	16.5		47.7
20	.833	.0328	10.00	79.67	10.3		58.0
35	.417	.0164	8.18	87.85	10.7		68.7
65	.208	.0082	4.56	92.41	10.2		78.9
100	.147	.0058	1.98	94.39	6.1		85.0
150	.104	.0041	1.17	95.56	4.1		89.1
200	.074	.0029	2.03	97.59	5.7		94.8
<200	.074	.0029	2.41	100.00	5.2		100.0
Total or average.....			100.00		2.47	100.00	

CHATS FROM ROUGHER JIG NO. 1.

3	6.680	0.263	7.79	4.00	7.37		
6	3.327	.131	27.27	35.06	47.41		54.78
10	1.651	.065	27.18	62.24	22.18		76.96
20	.833	.0328	12.42	74.66	5.44		82.40
35	.417	.0164	15.24	89.90	5.05		87.45
65	.208	.0082	5.97	95.87	3.32		90.77
100	.147	.0058	1.60	97.47	2.39		93.16
150	.104	.0041	.56	98.03	1.52		94.68
200	.074	.0029	.46	98.49	1.27		95.95
<200	.074	.0029	1.51	100.00	4.05		100.00
Total or average.....			100.00		4.23	100.00	

TABLE 19.—*Results of screen analyses of feeds and products of rougher and cleaner jigs—Continued.*

CHATS FROM ROUGHER JIG NO. 2.

Screen.			Weight of screen products.	Cumulative weight.	Zinc content by assay.	Percentage of total zinc.	Cumulative zinc content.
Size of mesh.	Size of opening.						
	Mm.	Inches.					
3	6.680	0.263	<i>Per cent.</i> 5.98	<i>Per cent.</i> 22.80	<i>Per cent.</i> 1.19	2.8	<i>Per cent.</i> 18.0
6	3.327	.131	16.82	22.80	2.25	15.2	18.0
10	1.651	.065	29.15	51.95	2.85	33.4	51.4
20	.833	.0328	23.09	75.04	1.91	17.8	69.2
35	.417	.0164	15.85	90.89	1.60	10.2	79.4
65	.208	.0082	5.95	96.84	2.37	5.7	85.1
100	.147	.0058	1.34	98.18	9.21	5.0	90.1
150	.104	.0041	.13	98.31	9.50	.5	90.6
200	.074	.0029	.75	99.06	17.40	5.3	95.9
<200	.074	.0029	.94	100.00	10.80	4.1	100.0
Total or average.....			100.00		2.48	100.00	

CHATS FROM CLEANER JIG.

6	3.327	0.131	8.82		28.80	19.98	
10	1.651	.065	19.45	28.27	10.45	15.99	35.97
20	.833	.0328	20.38	48.65	7.70	12.35	48.32
35	.417	.0164	27.41	76.06	10.05	21.67	69.99
65	.208	.0082	17.05	93.11	12.80	17.17	87.16
100	.147	.0058	4.18	97.29	23.00	7.57	94.73
150	.104	.0041	.41	97.70	26.50	.96	95.59
200	.074	.0029	1.34	99.04	27.45	2.89	98.48
<200	.074	.0029	.96	100.00	20.10	1.52	100.00
Total or average.....			100.00		12.71	100.00	

SPIGOT TAILING FROM CLEANER JIG.

6	3.327	0.131	1.62		12.40	2.0	
10	1.651	.065	14.70	16.32	7.83	11.3	13.3
20	.833	.0328	21.84	38.16	6.75	14.5	27.8
35	.417	.0164	37.85	76.01	7.30	29.45	57.25
65	.208	.0082	16.63	92.64	14.05	23.0	80.25
100	.147	.0058	4.39	97.03	29.10	12.55	92.80
150	.104	.0041	.69	97.72	28.80	1.95	94.75
200	.074	.0029	1.40	99.12	27.30	3.75	98.50
<200	.074	.0029	.88	100.00	17.40	1.5	100.00
Total or average.....			100.00		10.15	100.00	

TOTAL CHATS FROM JIGS.

3	6.680	0.263	4.47		3.82	3.0	
6	3.327	.131	15.89	20.36	6.45	18.2	21.2
10	1.651	.065	24.74	45.10	5.05	22.2	43.4
20	.833	.0328	20.64	65.74	3.30	12.1	55.5
35	.417	.0164	22.10	87.84	4.75	18.7	74.2
65	.208	.0082	8.16	96.00	8.50	12.3	86.5
100	.147	.0058	2.02	98.02	22.60	8.1	94.6
150	.104	.0041	.40	98.42	19.80	1.4	96.0
200	.074	.0029	.77	99.19	16.40	2.3	98.3
<200	.074	.0029	.81	100.00	11.80	1.7	100.0
Total or average.....			100.00		5.63	100.0	

All of the screen analyses and assays in Table 19 are on feeds and products from the same mill. The tailing from the cleaner jig is included in the total chats from all the jigs.

Other points in favor of treating the recrushed chats over chat jigs is that returning them to the rougher jig would (1) increase the quantity of fines and slimes in the feed to that jig, and (2) when the quantity of chats is relatively large, reduce its capacity to a certain extent, and (3) increase the chance for loss in the finest sizes. This also brings out the advisability of desliming at the head of the rougher jig, the reasons for which have already been given.

CLEANER JIGS.

The efficiency of cleaner jigs used in the mills of this district varies from 75 to 85 per cent. At the end of the jig there is usually a dewatering box similar to the kind used at the end of rougher jigs where dewatering screens are not used. The tailing is discharged from the box through a spigot and the overflow passes out into settling tanks. The tailing is seldom discarded, being usually reground for further treatment.

SUGGESTIONS FOR IMPROVEMENT OF JIGGING PRACTICE.

Based on the preceding data the chief suggestions offered for jigging practice in the mills of the Joplin district are as follows: (1) Desliming at the head of the rougher jigs; (2) reducing the quantity of unnecessary top water on the rougher jigs; and (3) retreating recrushed chats over separate chat jigs or tables by the use of a roughing and cleaning system.

Another suggestion that might be added is the use of two or more rougher jigs in mills treating 200 tons or more in 10 hours. In many of the mills visited, the addition of a second rougher jig would seem advisable to prevent the crowding caused by forcing the entire feed over one rougher jig and thus reducing its efficiency, to permit a better separation of the mineral from the gangue, and to have a relatively larger capacity available whenever it should be found necessary. Also, by having two rougher jigs the quantity of water used per ton of ore and the effective action of the jig could be more readily controlled.

CONCENTRATION OF FINE MATERIAL.

During the past few years there has been a marked improvement in the treatment of the finer material of the lead and zinc ores in this district. A few years ago tables were seldom used, but now nearly every mill contains 3 to 10 tables, and in a few of the more recent mills 20 or more tables are used. The small number of tables generally used is due to the small tonnage of ore treated, the capacity of the plants ranging from 100 to 500 tons in 10 hours, and the attempt to minimize the quantity of fine material. The aim in the following

pages is to emphasize the more important factors necessary to good table practice and to show possibilities of improving the average practice in the district.

OUTLINE OF PRACTICE IN THE TREATMENT OF FINE MATERIAL.

The feed to the table section, locally known as the "sludge" section, of any mill in this district comes chiefly from the dewatering screens and overflow waters from dewatering boxes at the end of jigs. In a few mills a large proportion of the finer sands and slimes is eliminated from the feed to the jigs, being diverted by launders to the settling tanks. The type of settling tank commonly used is rectangular, about 12 feet wide, 20 to 25 feet long, 2 to 3 feet deep at one end, and 6 to 8 feet deep at the other end, with the bottom sloping toward one corner of the tank. In most mills the material is fed into the settling tank at the deep end and overflows at the shallow end, although at a few plants the feed enters the shallow end and overflows at the deep end. Some operators are now using double tanks in which the overflow at the shallow end of the first tank flows into the shallow end of a second similar tank, which is placed parallel to the first tank, and discharges at the deep end of the second tank. The double-tank system gives the material a double settling and a rough classification of coarse and fine sands. When the settling tanks, or first tank where the double-tank system is used, has become about two-thirds full of sand, the flow or feed is diverted into another tank or set of tanks, while the material in the filled tank is discharged to the table section of the mill.

Recently Dorr thickeners have been installed at several mills; in general their purpose has been chiefly to settle and thicken slimes, although at a few mills they have been used for treating the coarser sands in order to give a more uniform feed to the tables.

At most mills the feed to the settling tanks enters a V-box with a continuous discharge, thus eliminating much of the coarser sand from the feed overflowing at the top of the box into the settling tank, and incidentally increasing somewhat the settling capacity of the large tanks.

The material discharged from both the V-box and the settling tank is raised by an elevator and discharged onto a trommel. The undersize from the trommel passes through classifiers and thence to tables; the oversize is either discarded as waste or crushed by rolls and returned to the trommel in closed circuit.

FINE SCREENING.

Trommels are most commonly used for screening in the table section of the mills. In a few mills, however, a local type of flat screen known as the Henry screen is used.

USE OF TROMMEL SCREENS.

The efficiency of trommels for screening fine material depends on the same conditions that were given for coarse screening. In general the fine trommel screens are less efficient than the coarser screens, because the openings have more of a tendency to blind and the finer feed particles tend to cling together in small balls on the screen as soon as they become more or less dewatered. The efficiency of trommels for screening the fines from certain products in the table sections of mills throughout the district varies considerably, as was found from a number of tests. Two examples are given to show both poor and good results.

TEST 1.—SCREENING EFFICIENCY OF TROMMEL WITH 1-MM. OPENINGS; SIZE, 48×72 INCHES; SPEED, 14 REVOLUTIONS PER MINUTE; SLOPE, 1½ INCHES PER FOOT.

	Trommel feed.	Trommel oversize.
Oversize on 1-mm. screen, per cent.....	16.1	28.8
Undersize through 1-mm. screen, per cent.....	83.9	71.2
	100.0	100.0

$$\text{Weight of undersize} = \frac{16.1 \times 71.2}{28.8} = 39.8$$

$$\text{Efficiency} = \frac{\text{Weight of undersize going through the screen}}{\text{Total weight of undersize in feed to screen}} =$$

$$\frac{(83.9 - 39.8) \times 100}{83.9} = 52.56 \text{ per cent.}$$

TEST 2.—SCREENING EFFICIENCY OF TROMMEL WITH 1-MM. OPENINGS; SIZE, 48×72 INCHES; SPEED, 16 REVOLUTIONS PER MINUTE; SLOPE, 1½ INCHES PER FOOT.

	Trommel feed.	Trommel oversize.
Oversize on 1-mm. screen, per cent.....	28.5	63.7
Undersize on 1-mm. screen, per cent.....	71.5	36.3
	100.0	100.0

$$\text{Weight of undersize} = \frac{28.5 \times 36.3}{63.7} = 16.4.$$

$$\text{Efficiency} = \frac{\text{Weight of undersize passing through screen}}{\text{Total weight of undersize in feed to screen}} =$$

$$\frac{(71.5 - 16.4) \times 100}{71.5} = 77.0 \text{ per cent.}$$

The results in the first example show that the rate of speed of the trommel was somewhat slow and that the slope was not sufficient for the speed. It was noticed while the trommel was running that the feed bedded thickly on the screen, preventing a certain proportion of the undersize material from working through the openings. Later this thick bedding was largely remedied by increasing the speed to

about 17 revolutions per minute and the slope to $1\frac{1}{2}$ inches per foot. These changes also increased the screening efficiency of the trommel.

In mills where the oversize from sand trommels is discarded as tailing, screen analyses of this oversize should be made and the various screen products assayed to determine in what sizes the mineral values are and what portion is possibly worth saving. The distribution of zinc in an oversize from a sand trommel is shown by the screen analysis and assays in Table 20 following:

TABLE 20.—*Results of screen analysis and assays of oversize from sand trommel.*

Screen.			Weight of screen products.	Cumulative weight.	Zinc content.		
Size of mesh.	Size of openings.				By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
			<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>		
20	0.833	0.0328	45.06		1.70	52.0	
35	.417	.0164	40.87	85.93	.75	20.8	72.8
65	.208	.0082	9.26	95.19	1.55	9.75	82.55
100	.147	.0058	2.36	97.55	4.85	7.75	90.30
150	.104	.0041	.22	97.77	4.85	.75	91.05
200	.074	.0029	1.07	98.84	5.15	3.75	94.80
<200	.074	.0029	1.16	100.00	6.65	5.20	100.00
Total or average.....			100.00		1.47	100.00	

The results show that the zinc content of the oversize was 1.47 per cent zinc. The oversize on the 20-mesh screen assayed 1.70 per cent zinc, whereas the material between 20 and 35 mesh assayed only 0.75 per cent zinc. This indicates that the larger zinc content in the material coarser than 20 mesh was probably due to the presence of particles containing included zinc mineral, or chats, that would have to be reground before further treatment. With all material coarser than 35 mesh eliminated from consideration, let the value of that passing the 35-mesh screen be considered. The calculated zinc content of this undersize is 2.92 per cent, or practically double that of the total oversize. For example, assume that 5 tons of this undersize can be screened from the oversize each day. With an assay value of 2.92 per cent zinc, this quantity would contain 0.146 ton of zinc, or, in terms of 50 per cent zinc concentrates, 0.292 ton. For every 5 tons of this material treated on tables, with a 65 per cent zinc recovery, practically 0.19 ton of 50 per cent zinc concentrate would be saved. At \$50 a ton for this product the value of the quantity saved each day would be \$9.50. This calculation is simply to show how a profitable, although small, saving can be effected by more efficient screening. In this mill, however, the writer believes that it would have paid to regrind the entire oversize and treat it over the tables with the main table feed.

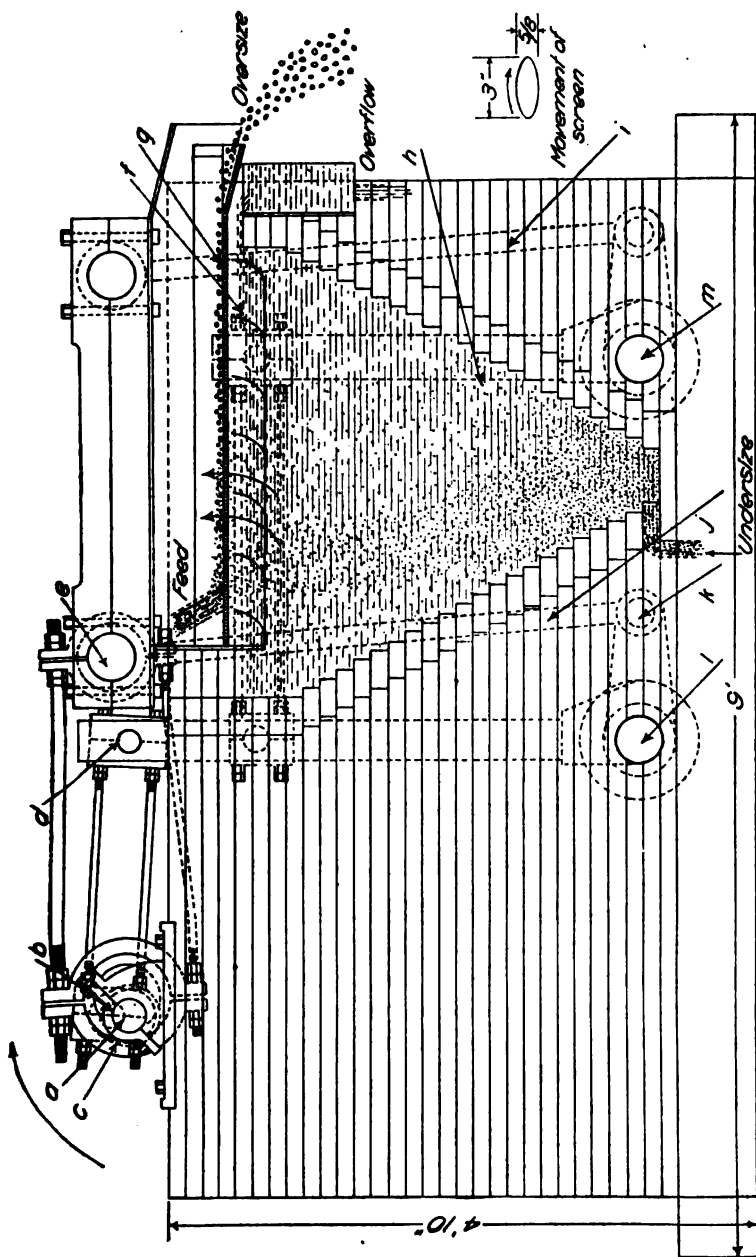


FIGURE 11.—Sketch of Henry screen. *a*, main shaft; *b*, vertical motion crank; *c*, horizontal motion eccentric; *d*, quadrant; *e*, connecting box; *f*, blades; *g*, screen; *h*, hutch; *i, j*, carrier legs; *k*, quadrant; *l*, rear rocker shaft; *m*, front rocker shaft.

THE HENRY SCREEN.

The Henry screen is shown in figure 11. The main frame carries the mechanism that transmits movement to the screen. Within the frame is a hopper-shaped tank, with ample pitch to prevent the fine material adhering to the sides. The driving shaft carries two eccentrics. One of these connects directly to the screen carrier and imparts to the screen a forward-and-backward movement. The second eccentric leads the first 90° , and by means of arms gives the screen an up-and-down movement. The mean of these two movements combined is a rising forward movement during one-half of the revolution of the shaft and a downward backward movement during the other half of the revolution. The material is fed into the rear end of the screen. The resulting action of the screen from each revolution tends to carry the oversize to the forward end, where it is discharged; the undersize passes through the screen into the tank or hutch, where it is drawn off at the bottom by means of a spigot plug.

The water that carries the pulp to the screen passes through the screen into the tank, where its level is maintained, by means of an adjustable gate, at a point below the level of the screen when it is at midstroke. The blades supporting the screen are securely joined to both sides of the screen carrier.

These blades are so shaped that during the downward backward movement of the screen carrier they dip into the water and force it up against the screen with a pulsating effect, thus cleaning the openings of the screen and assisting the forward movement of the oversize particles. As the screen lifts forward from the water it washes the fines through the openings and cleans the coarse sand of adhering particles.

CLASSIFICATION OF FINES.

In the general practice throughout the district, the undersize from the sand trommel passes into classifiers, usually of the ordinary V-box type with water under hydraulic pressure passing up through the sorting column, and discharges onto the tables. The writer believes that separation of the slimes from the sands before classification would be a decided advantage, both in increasing the efficiency of the classifiers and in decreasing the losses in slimes. For desliming, one of the well-known types of desliming classifiers, such as the Dorr, the Akins, or the Federal-Esperanza or some similar type of drag classifier could be used. Both the Dorr and the Akins have been tried in some of the larger mills with satisfactory results, and it is believed that any of the types of deslimers mentioned would greatly increase the efficiency of the table section of any mill. Not only would a better classification of the feeds to the various tables be obtained, but the proportion of water to solids could be more readily

regulated for the proper thickness of pulp desired for each table. Also by eliminating the larger proportion of the slimes from the material fed to classifiers these can be more easily adjusted. The pulp which could go direct to some settling tank for thickening, if necessary, would be much thicker than when it passes over the various classifiers and becomes diluted with water from each classifier so as to require a greater ratio of thickening before treatment on subsequent slime tables or by flotation.

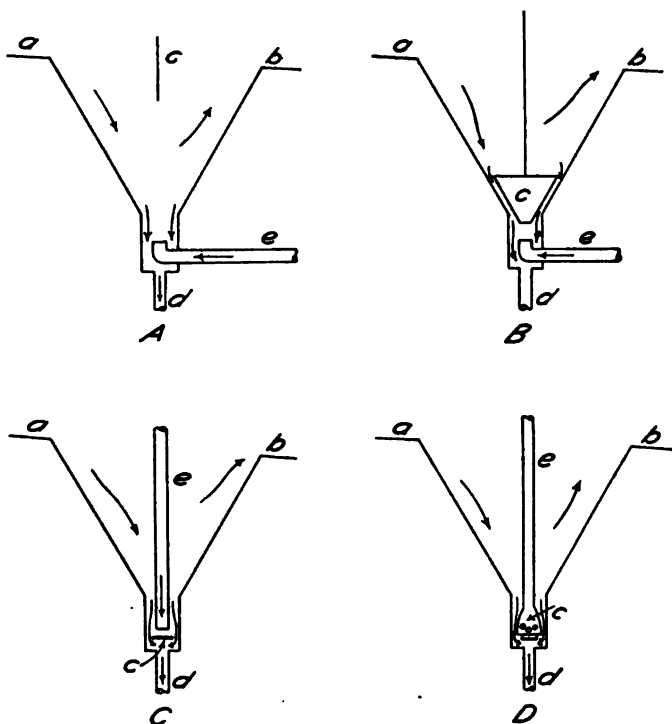


FIGURE 12.—V-type classifiers.

TYPES OF CLASSIFIERS USED.

As stated previously, the usual type of classifier used is a series of V-boxes with hydraulic flow, placed in consecutive order above each table, the overflow from one box flowing through a launder to the next box. The hydraulic water enters each box at a point near the bottom. Different methods of feeding water and pulp are shown in figure 12. In type A, figure 12, the pulp is fed into the classifier at *a*. The relatively coarser and heavier particles drop into the sorting column, through the current of water rising from pipe *e*, and pass through the discharge pipe *d* onto the table. The lighter particles pass under the baffle *c* and overflow at *b*, passing through the launder to the next classifier.

The baffle *c* is placed at right angles to the flow of the feed to retard the speed of the flowing current. Most of the classifiers of the

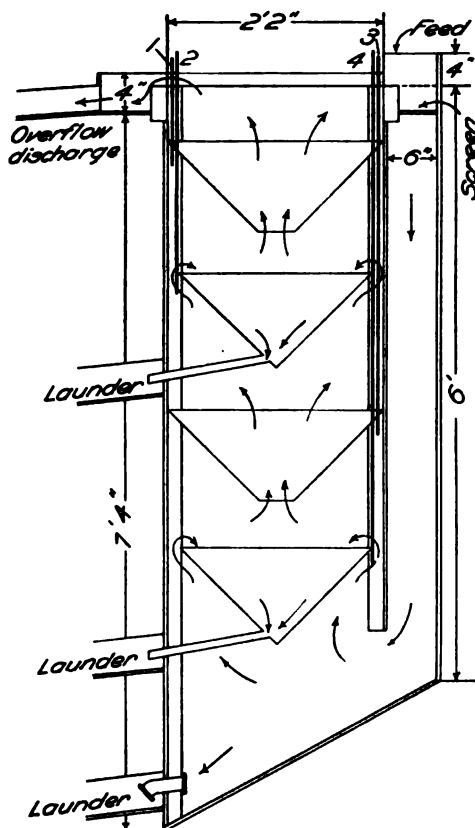
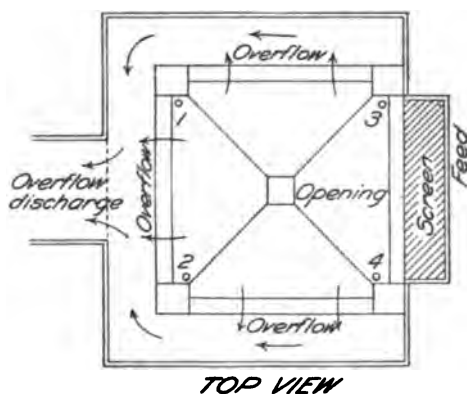


FIGURE 13.—Bird classifier.

Type *B*, figure 12, is the same as type *A*, except that a wooden block *c*, shaped like the frustum of a pyramid, is placed in inverted position at the head of the sorting column, the four sides being parallel with and equidistant from the sloping sides of the classifier. The pulp, as it drops down through the narrow passage thus formed, meets the rising current of water from the pipe *e*; hence all the particles discharged through *d* must pass through this upward flow. The block *c* is fastened to a vertical shaft that can be raised or lowered, thus varying the width of the aperture. The aperture, measured at right angles to the sides, is usually $\frac{1}{4}$ inch to $\frac{3}{4}$ inch wide, depending on the size of the material passing into the classifier.

In *C*, figure 12, the water is forced in through a vertical pipe *e* from the top, and not through a horizontal pipe from the side, as in the first two classifiers described. As the water discharges under pressure at the lower end of the vertical pipe *e* it strikes against

a horizontal disk or plate *c* of the same shape as the cross section of the sorting column, and consequently tends to rebound up through the sorting column against the downward flow of the pulp. This disk

or plate is so placed that its edges are equidistant from the sides of the sorting column, forming an opening $\frac{1}{2}$ to $\frac{3}{4}$ inch wide. The particles pass through this aperture and discharge at *d*.

The classifier represented in D, figure 12, is found in only a few of the mills. The vertical pipe *e* terminates in a bell that is closed at the end and has around the sides small circular openings, about one-fourth inch in diameter, through which the water passes. The action of the water is about the same as in classifier C, except that the flow is divided by the circular openings into small streams. This type of classifier is said to give good results.

Classifiers of the Richards-Janney type are found in a few mills; also, a local machine known as the Bird classifier. A self-explanatory sketch of this classifier is given in figure 13. The vertical $\frac{1}{2}$ -inch air pipes, 1, 2, 3, and 4, extend into the different parts of the classifier to prevent any disturbance resulting from the formation of air bubbles as the pulp and water rise through the classifier, the air passing out through these pipes.

RESULTS OF SCREEN TESTS OF CLASSIFIER PRODUCTS.

The results of screen tests of feed products to tables from classifiers of the V-box type are shown in Table 21, following. The classifiers all used hydraulic flow, except classifiers 8 and 9. Launderers connected the nine classifiers, and each classifier delivered to a separate table. The dimensions of each classifier and the size of the spigot-discharge opening are given in Table 22.

TABLE 21.—Results of screen tests of spigot products from nine classifiers fed to tables.

Size of mesh.	Feed to classifiers.		Underflow from classifier 1.		Underflow from classifier 2.		Underflow from classifier 3.	
	Weight.	Cumulative weight.	Weight.	Cumulative weight.	Weight.	Cumulative weight.	Weight.	Cumulative weight.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
On 20	0.36	0.36	0.79	0.79	0.73	0.73	16.12	16.12
35	15.03	15.39	25.85	26.64	21.49	22.22	40.26	56.38
65	32.14	47.53	38.49	65.13	42.99	65.21	18.24	74.62
100	18.98	66.51	15.75	80.88	15.67	80.88	14.20	88.81
150	18.71	85.22	10.78	91.66	11.27	92.15	4.95	93.76
200	6.52	91.74	3.60	95.26	3.59	95.74	6.13	99.89
Through 200	8.27	100.01	4.72	99.98	4.25	99.99		
Total....	100.01		99.98		99.98		99.99	

Size of mesh.	Underflow from classifier 4.		Underflow from classifier 5.		Underflow from classifier 6.		Underflow from classifier 7.	
	Weight.	Cumulative weight.	Weight.	Cumulative weight.	Weight.	Cumulative weight.	Weight.	Cumulative weight.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
On 20	9.32	9.32	3.17	3.17	0.69	0.69	0.23	0.23
35	35.41	14.73	25.60	28.77	11.24	11.93	5.35	5.58
100	21.46	66.19	26.95	55.73	26.17	38.10	16.41	21.99
150	18.25	84.44	24.62	80.35	34.18	72.28	33.84	55.83
200	7.58	92.02	10.00	90.35	13.36	85.64	16.55	72.38
Through 200	7.95	99.97	9.63	99.98	14.42	100.06	27.61	99.99
Total....	99.97		99.98		100.06		99.99	

TABLE 21.—Results of screen tests of spigot products from nine classifiers fed to tables—Continued.

Size of mesh.	Underflow from classifier 8.		Underflow from classifier 9.		Overflow from classifier 9.	
	Weight.	Cumulative weight.	Weight.	Cumulative weight.	Weight.	Cumulative weight.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
On 20						
35	0.10	0.10			3.81	3.81
65	1.33	1.43	1.04	1.04	0.14	3.95
100	7.34	8.77	4.78	5.82	4.52	8.47
150	24.75	33.52	26.95	32.77	4.38	12.85
200	21.90	55.42	20.31	53.08	2.97	15.82
Through 200	44.57	99.99	46.91	99.99	84.17	99.99
Total...	99.99		99.99		99.99	

TABLE 22.—Sizes of the nine classifiers.

Classifier number.	Size of top (square).	Depth.	Diameter of spigot-discharge opening.
	<i>Inches.</i>	<i>Inches.</i>	<i>Inches.</i>
1	16 by 16	24	1
2	21 by 21	26	$\frac{1}{2}$
3	22 by 22	30	$\frac{1}{2}$
4	28 by 28	36	$\frac{1}{2}$
5	30 by 30	46	$\frac{1}{2}$
6	35 by 35	46	$\frac{1}{2}$
7	41 by 41	54	$\frac{1}{2}$
8	48 by 48	54	$\frac{1}{2}$
9	45 by 45	54	$\frac{1}{2}$

TABLE PRACTICE.

ESSENTIAL FACTORS IN TABLE PRACTICE.

FOUNDATIONS.

In ordinary practice the deck of a table has a speed of 200 to 300 strokes a minute. The table surface should be free from vibrations that make ripples in the water and in the table feed, for these cause losses. Hence the table should have a firm foundation. The best support is a solid concrete foundation extending the entire length. In the Joplin district a concrete pier under the head and another under the opposite end of the table have proved satisfactory. These piers have wide bases. A concrete pier under the head only is usually not firm enough, and a table bolted directly to the floor of the mill is bound to vibrate. For best results, the table supports must be rigid and securely bolted to a firm foundation, free from the floor of the mill, unless the floor is of concrete.

HEAD MOTION.

The head motion advances the ore or pulp along the deck of the table, the speed, or number of strokes per minute, being determined by the length of stroke and the size of material treated. Results obtained with various tables in the district indicated that the best

speed for classified material passing a $1\frac{1}{2}$ -mm. screen was 220 to 240 strokes a minute with a 1-inch to $\frac{3}{4}$ -inch stroke for the coarser sands, 240 to 260 strokes a minute with a $\frac{3}{4}$ -inch to $\frac{1}{2}$ -inch stroke for intermediate sizes, and 260 to 300 strokes a minute with a $\frac{1}{2}$ -inch to $\frac{1}{4}$ -inch stroke for the finer sizes. The effectiveness of these speeds and lengths of stroke, however, depends on the slope of the table, the amount of wash water used, the quantity of feed, and the character of the ore treated.

FEED TO TABLES.

Perhaps the most important factor in successful table concentration is uniform feed. If the feed is uniform and continuous, the lines of separation vary less, and the quantity of wash water and the slope of the table can be much more easily regulated. To insure a uniform feed, the material fed must be well classified. In the mills of the Joplin district sands or fine materials are usually classified in simple V-shaped boxes, with or without baffles, a column of water under pressure passing through the discharge end or sorting column. The overflow passes over a launder, which has a slight slope, to the next box, the depth of the boxes increasing with the fineness of the material. The proper thickness of pulp or the quantity of water in the feed to a table, especially for the finer sizes, must be determined by the mill foreman.

SLOPE OF TABLES AND QUANTITY OF WASH WATER.

The quantity of wash water and the slope given to a table should be regulated according the quantity and size of the feed. Coarse sand requires more wash water than a fine pulp. Tilting of the table is used more especially for adjusting the line of separation when the feed varies. The function of the wash water is to clean the concentrate before it leaves the table.

PRODUCTS FROM TABLES.

As the specific gravity of galena (7.4 to 7.6) and of zinc blende (3.9 to 4.1) is decidedly higher than that of flint (2.65), separating the lead and zinc minerals from the flint in the coarse and intermediate size sands by treatment on tables is not difficult, but effecting such separation when the pulp consists mostly of slimes or material passing a 200-mesh screen is not so easy. The iron sulphide, which is usually marcasite (specific gravity 4.85 to 4.9), is difficult to separate from the zinc blende in either coarse or fine feeds, because the difference in specific gravity of the two minerals is small.

A clean lead concentrate and a clean zinc concentrate are usually made, the intermediate mixed lead, iron, and zinc concentrate, known as the "return," being returned in most mills to the original table feed.

The middling is the product between the concentrate and the tailing. Although there are exceptions, the usual practice in the Joplin

mills is to return the middlings from the tables to the original table feed without regrinding.

The lead and zinc content of the tailings depends on the richness of the feed to the table, the classification and uniformity of the feed, the action of the table, and the size of the material. As a rule, the finer the material treated, the higher will be the assay value of the tailing.

The personal attention given to tables is an important factor in obtaining a high recovery. Tables are by no means "fool proof" and are very sensitive to variations of feed.

RESULTS OF TABLE TESTS.

Results of several table tests in various mills are given to show the character of the table practice in the district. The variations in zinc recovery are partly caused by the factors already enumerated. In all of the tables the zinc content is given in terms of metallic zinc.

Table 23 shows that when the ore fed to tables is more or less "chatty," the zinc mineral not having been entirely liberated from the gangue by crushing, the percentage of recovery is lower than when the ore is "free"—that is, when the zinc mineral breaks cleanly from the gangue when crushed to table size. It is believed that the percentage of recovery from this chatty ore could be raised by regrinding the middlings from the tables, a point that is brought out more fully on a subsequent page.

TABLE 23.—Results of tests of table sections in various mills.

Test No.	Number of tables in operation.	Zinc content of feed to tables.	Zinc content of concentrates produced from tables.	Zinc content of tailings from tables.	Percentage of zinc recovered.	Character of ore treated.
		<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>		
1	1	6.35	58.0	3.75	43.8	Chatty.
2	1	4.90	59.2	1.50	71.2	Free.
3	2	2.95	55.6	1.35	55.5	Chatty.
4	2	4.85	57.4	1.70	67.0	Do.
5	2	4.70	57.0	1.55	69.0	Do.
6	2	3.12	54.5	1.00	69.1	Do.
7	2	4.30	56.2	1.35	70.2	Do.
8	2	5.65	57.1	1.62	73.5	Free.
9	2	7.75	57.2	2.25	73.9	Do.
10	3	3.75	52.8	1.20	69.5	Chatty.
11	3	6.75	54.6	1.55	79.4	Free.
12	4	4.65	42.0	1.30	74.3	Chatty.
13	6	3.95	59.3	1.19	71.3	Free.
14	7	4.65	55.0	1.42	71.3	Chatty.
15	9	5.25	52.1	1.00	82.5	Chatty-free.
16	10	4.90	57.5	1.01	80.8	Do.

EFFECT OF SPEED AND LENGTH OF STROKE.

The best speed and length of stroke for a table must be determined by the mill foreman; they will depend on the character of the feed.

To show the effect of wrong speeds and wrong lengths of stroke the results of two tests are given in Table 24 following. The feed passed a 1½-mm. screen.

TABLE 24.—Results of tests showing wrong speeds and lengths of stroke.

	Test 1.	Test 2.
Speed of table, strokes per minute.....	175	224
Length of stroke, inches.....	1½	½
Zinc content of feed, per cent.....	9.25	7.75
Zinc content of concentrates produced, per cent.....	52.2	57.2
Zinc content of tailings, per cent.....	3.05	3.20
Percentage of zinc recovered.....	71.3	62.2

The percentages of zinc recovery in Table 24 are poorer than the results indicate, for the feed was considerably richer than the average feed to tables, and should have given, in both tests, much higher recoveries with proper operation. In test 1 the speed of the table was too slow and the stroke too long for the size of material treated; in test 2 the stroke was too short for the speed of the table and the size of material treated.

TESTS OF FEEDS AND PRODUCTS FROM SIX TABLES TREATING "FREE" SHEET-GROUND ORE.

In order to determine the actual efficiency of each table in a mill and to find just where losses are taking place, it is necessary to make screen analyses of the feed and table products. To show the importance of making such screen analyses a few results are given.

Tables 25 and 26 show the results of treating a sheet-ground ore over six tables, and Tables 27 and 28 of treating a "chatty" ore over four tables. In both mills the middlings and mixed "return" concentrate products of lead, iron, and zinc from all tables were returned to the original table feed without being reground.

TABLE 25.—Results of tests with six concentrating tables treating "sheet-ground" ore.

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Mm.	Inch.	Percent- age of weight.	Zinc assay, per cent.	Percent- age of total zinc content.	Percent- age of weight.	Zinc assay, per cent.	Percent- age of total zinc content.
Table 1 (speed, 260; ½-inch stroke).....	20	0.833	0.0328						
	35	.417	.0164	22.75	0.21	1.85	30.80	0.20	7.35
	65	.208	.0082	38.55	.54	7.90	41.95	.36	13.00
	100	.147	.0058	16.60	2.52	14.60	12.35	.72	10.60
	150	.104	.0041	10.65	6.76	25.30	6.45	1.61	12.35
	200	.074	.0029	1.15	7.87	3.45	1.20	2.31	3.30
	<200	.074	.0029	10.30	12.01	46.90	7.25	6.18	53.40
Total or average.....				100.00	2.64	100.00	100.00	.84	100.00

TABLE 25.—Results of tests with six concentrating tables treating "sheet-ground" ore—Continued.

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Mm.	Inch.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
Table 2 (speed, 260; $\frac{1}{2}$ -inch stroke).....	20								
	35	0.417	0.0164	16.70	0.21	1.12	19.7	0.13	2.60
	65	.208	.0082	41.20	.67	8.67	46.6	.24	11.30
	100	.147	.0058	18.50	2.86	16.98	11.8	.46	5.50
	150	.104	.0041	11.55	6.73	24.98	10.1	1.19	12.10
	200	.074	.0029	7.75	8.47	2.10	1.2	1.61	1.95
	<200	.074	.0029	11.30	12.66	45.95	10.6	6.26	66.55
Total or average..				100.00	3.11	100.00	100.0	.99	100.00
Table 3 (speed, 240; $\frac{1}{2}$ -inch stroke).....	20								
	35	0.417	0.0164	6.75	0.23	0.46	8.5	0.15	1.67
	65	.208	.0082	36.95	.39	4.36	38.5	.18	9.18
	100	.147	.0058	27.00	1.33	10.88	30.3	.44	17.60
	150	.104	.0041	17.80	5.98	32.19	12.9	1.49	25.44
	200	.074	.0029	1.80	12.32	6.72	3.8	1.98	9.92
	<200	.074	.0029	9.70	15.49	45.39	6.0	4.57	36.19
Total or average..				100.00	3.31	100.00	100.0	.76	100.00
Table 4 (speed, 230; $\frac{1}{2}$ -inch stroke).....	35	0.417	0.0164						
	65	.208	.0082	0.25	1.14	0.05			
	100	.147	.0058	9.60	.57	.92	15.2	0.34	3.32
	150	.104	.0041	38.70	1.25	8.79	44.45	.62	17.83
	200	.074	.0029	4.95	2.67	2.22	5.85	1.12	4.23
	<200	.074	.0029	46.50	11.28	83.02	34.50	3.35	74.62
Total or average..				100.00	5.95	100.00	100.00	1.55	100.00
Table 5 (speed, 300; $\frac{1}{2}$ -inch stroke).....	65	0.208	0.0082						
	100	.147	.0058	8.05	1.38	1.63			
	150	.104	.0041	29.85	2.13	9.29	32.0	0.73	6.35
	200	.074	.0029	6.45	3.33	3.14	9.7	1.12	2.94
	<200	.074	.0029	55.65	10.56	85.94	58.3	5.72	90.71
				100.00	6.84	100.00	100.0	3.68	100.00
Table 6 (speed, 300; $\frac{1}{2}$ -inch stroke).....	100	0.147	0.0058						
	150	.104	.0041	5.5	1.19	1.01	4.05	0.60	0.67
	200	.074	.0029	3.3	1.51	.75	2.70	.62	.45
	<200	.074	.0029	91.2	6.94	98.24	98.25	3.85	98.88
Total or average..				100.0	6.48	100.00	100.00	3.63	100.00

SUMMARY.

	Speed, strokes per minute.	Length of stroke.	Zinc content of feed.	Zinc content of concentrate.	Zinc content of tailing.	Percentage of zinc recovery.
		Inch.	Per cent.	Per cent.	Per cent.	
Table 1.....	260	$\frac{1}{2}$	2.64	61.15	0.84	69.1
Table 2.....	260	$\frac{1}{2}$	3.11	59.59	.99	69.3
Table 3.....	240	$\frac{1}{2}$	3.31	59.38	.76	78.1
Table 4.....	230	$\frac{1}{2}$	5.95	57.10	1.55	78.1
Table 5.....	300	$\frac{1}{2}$	6.84	57.98	3.68	49.3
Table 6.....	300	$\frac{1}{2}$	6.48	56.68	3.63	47.0
Total.....			3.95	59.3	1.19	71.3

DISCUSSION OF RESULTS WITH EACH OF THE SIX TABLES.

Concentrating table 1 treated the coarsest material; its feed contained 2.64 per cent zinc. The screen product between 35 and 65 mesh represented the largest proportion of material treated, or 38.55 per cent of the feed, and contained only 0.54 per cent zinc, or only 7.90 per cent of the total zinc content in the feed to the table. The screen analysis shows that 10.3 per cent of the feed passed the 200-mesh screen, which indicates poor classification. The material finer than 200-mesh contained 12.01 per cent zinc, or 46.90 per cent of the total zinc content in the feed. Evidently a considerable proportion of the zinc in this material passed into the tailings because the table was not adjusted for treating such fine material. The screen product of tailing finer than 200-mesh assayed 6.18 per cent zinc, and represented 53.40 per cent of the total zinc content in the tailing. The total tailing from this table assayed 0.84 per cent zinc, but would have contained considerably less with better classification. The percentage of zinc recovered from this table was 69.1 per cent, which is low for the size of material the table was supposed to treat. The feed was not uniform and the table was crowded at times.

Poor classification of the feed to table 2 is shown by the percentage of slimes present. The largest proportion of zinc loss in the tailing, or 66.55 per cent, was in the material passing the 200-mesh screen. The feed was not uniform.

Table 3 shows, relatively, better classification and a consequently higher zinc recovery. The percentage of material passing the 200-mesh screen was 9.70 per cent, or relatively much less than for tables 1 and 2, as this table was treating the material from the third classifier. The recovery from the finer sizes was also higher, as indicated by the differences in assay value of the corresponding screen products between the feed and tailing.

The recovery with table 4 was good, especially for the material passing the 200-mesh screen. The speed of the table, 230 strokes per minute, was too slow for the size of material that was being treated.

Tables 5 and 6 were supposed to be treating slimes or material finer than 200-mesh. However, table 5 was receiving a feed of which 44.35 per cent would not pass the 200-mesh screen, and table 6 was receiving a fairly uniform slime of which 91.2 per cent was finer than 200-mesh. The respective low recoveries, however, were due rather to too much water in the feed and on the table than to the classification. The pulp was too thin for efficient work.

COMPARISON OF RESULTS WITH THE SIX TABLES.

The average results of the tests given in Table 25 show a total zinc recovery of 71.3 per cent from the six concentrating tables. This recovery is not very good, considering the character of the ore, which

was not "chatty." The poorer recoveries with the first two tables were probably due chiefly to poor classification, although the speeds and the lengths of stroke affected the quality of the work.

<i>Speeds and lengths of stroke of tables.</i>		Speed, strokes per minute.	Length of strokes, inches.
Table 1.....		260	$\frac{1}{2}$
Table 2.....		260	$\frac{1}{2}$
Table 3.....		240	$\frac{1}{2}$
Table 4.....		230	$\frac{1}{2}$
Table 5.....		300	$\frac{1}{2}$
Table 6.....		300	$\frac{1}{2}$

A comparison of the data in Table 25 shows that the length of stroke decreased as the size of material decreased. The lengths were approximately correct, but the speeds decreased in the same direction, which is contrary to good practice. If the speeds on the first four tables had been 230, 240, 250, and 260 or 270 per minute, respectively, with strokes of 1 inch, $\frac{7}{8}$ inch, $\frac{3}{4}$ inch, and $\frac{5}{8}$ inch, it is believed that the tables would have shown better recoveries. However, the low efficiency of the first two tables was chiefly caused by rather poor classification and the lack of uniform feed.

Although the results may not show high recoveries, they indicate some of the necessary factors of good table practice; thus, (1) tables 1 and 2 gave poor recoveries owing to poor classification, lack of uniform feed, and wrong speeds; (2) tables 3 and 4 gave better results because of better classification; and (3) tables 5 and 6 gave poor results because there was too much water in the feed, the pulp being too thin.

The screen analysis and assays of the total tailings from the six tables, given in Table 26, show that 15.5 per cent of the tailings passed the 200-mesh screen, and this part assayed 5.07 per cent zinc, representing 65.78 per cent of the total loss in the tailings. These figures indicate that the pulp of this size (less than 200-mesh) should be of the proper thickness when treated on slime tables to effect the best commercial saving, and that the classification should be such that the finer sizes of material are found on the tables treating fine material and not on the tables treating the coarse sands.

TABLE 26.—Results of analyses of total tailings from all six tables.

Size of mesh.	Size of openings.		Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
	Mm.	Inches.			
20	0.833	0.0328			
35	.417	.0184	16.9	0.36	5.03
65	.208	.0082	32.7	.21	5.74
100	.147	.0058	16.5	.47	6.50
150	.104	.0041	15.50	.99	12.80
200	.074	.0029	2.9	1.92	4.15
<200	.074	.0029	15.5	5.07	65.78
Total or average.....			100.0	1.19	100.00

TESTS OF FEEDS AND PRODUCTS OF FOUR TABLES TREATING "CHATTY" ORE IN
MIAMI DISTRICT.

In Table 27 the results represent the treatment of a "chatty" ore on four concentrating tables, not a "free" ore as in Table 25, as the zinc mineral was not entirely liberated from the gangue before treatment. This ore also carried much lead and iron sulphides that reduced the grade of the zinc concentrates. Moreover, the gangue material was entirely different, and the comparatively low zinc assays of the finer screen products were probably due to the gangue containing considerable "soapstone," which seemed to break into very fine pieces that mixed with the finer particles of ore.

TABLE 27.—Results of tests with four concentrating tables treating "chatty" ore.

(Miami District.)

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Milli-meters.	Inches.	Percent- age of weight.	Zinc assay, per cent.	Percent- age of total zinc content.	Percent- age of weight.	Zinc assay, per cent.	Percent- age of total zinc content.
Table 1 (speed, 252; 1-inch stroke).	10	1.651	0.065						
	20	.833	.0328	14.5	11.45	15.5	28.65	1.75	29.0
	35	.417	.0164	40.8	13.25	50.65	49.5	2.00	57.3
	65	.208	.0082	24.1	10.17	22.9	10.9	.90	5.65
	100	.147	.0058	9.4	6.67	5.9	3.3	.55	1.05
	150	.104	.0041	5.3	5.12	2.5	2.35	.70	.95
	200	.074	.0029	2.3	4.70	1.0	1.5	.95	.85
	<200	.074	.0029	3.6	4.45	1.55	3.8	2.35	5.2
Total or average..				100.0	10.65	100.00	100.00	1.73	100.00
Table 2 (speed, 236; 1½-inch stroke).	10	1.651	0.065						
	20	.833	.0328	0.5	2.65	0.4			
	35	.417	.0164	12.45	3.45	13.2	16.55	2.00	34.3
	65	.208	.0082	45.50	2.90	40.65	40.10	.65	27.0
	100	.147	.0055	12.60	3.25	12.6	20.70	.55	11.8
	150	.104	.0041	15.35	3.32	15.7	12.25	.52	6.6
	200	.074	.0029	7.35	4.00	9.05	4.75	1.22	6.0
	<200	.074	.0029	6.25	4.35	8.4	5.65	2.45	14.3
Total or average..				100.00	3.25	100.00	100.00	.96	100.0
Table 3 (speed, 245; 1½-inch stroke).	20	0.833	0.0328						
	35	.417	.0164	1.3	2.05	1.2	12.65	0.95	14.2
	65	.208	.0082	15.5	.95	6.9	17.35	.37	7.65
	100	.147	.0058	33.0	1.42	21.9	21.90	.37	9.65
	150	.104	.0041	10.4	1.85	9.0	21.20	.50	12.60
	200	.074	.0029	13.4	2.55	16.0	7.20	.65	5.60
	<200	.074	.0029	26.4	3.65	45.0	19.70	2.15	50.30
Total or average..				100.0	2.14	100.0	100.00	.84	100.00
Table 4 (speed, 245; 1-inch stroke).	35	0.417	0.0164						
	65	.208	.0082	3.0	0.40	0.7	4.0	0.30	1.4
	100	.147	.0058	21.6	.75	9.0	29.6	.40	14.0
	150	.104	.0041	30.1	1.15	19.2	21.4	.40	10.2
	200	.074	.0029	23.2	2.45	31.7	15.6	.70	13.0
	<200	.074	.0029	22.1	3.20	39.4	29.4	1.75	61.4
Total or average..				100.00	1.80	100.00	100.0	.84	100.0

TABLE 27.—*Results of tests with four concentrating tables testing "chatty" ore—Contd.*
SUMMARY.

Table.	Speed, strokes per minute.	Length of stroke.	Zinc content of feed.	Zinc con- tent of concent- rates.	Zinc content of tailing.	Percent- age of zinc recovery.
		<i>Inches.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	
Table 1.....	252	1	10.65	40.86	1.73	87.5
Table 2.....	236	1½	3.25	43.15	.96	72.0
Table 3.....	245	1½	2.14	43.74	.84	62.1
Table 4.....	245	1	1.80	42.11	.84	54.4
Total or average.....			4.65	42.00	1.30	74.3

DISCUSSION OF RESULTS WITH EACH TABLE.

The screen analysis of the feed to the first table shows a better classification than for the feed to the first table in the mill treating "free" ore. The assay values of the screen products are higher in the coarser sizes of material than in the finer, probably because a large quantity of middling and "return," the latter a mixed concentrate of lead, iron, and zinc, was being returned to the feed. The tailing products on the 20 and 35 mesh screens from this table assayed 1.75 and 2.00 per cent zinc respectively, whereas the zinc content of the next finer screen products was much lower. Evidently a considerable part of this middling needs to be reground before the zinc mineral in it can be recovered on tables, as a comparison of the 35-mesh screen products in the tailings from the first three tables, with the total tailings from all tables, will show. In each test the zinc content of the material coarser than 35 mesh is considerably higher than that of the screen products finer than 35 mesh, except in the material that passed the 200-mesh screen. Examination of this size material with a magnifying glass gave additional proof. In the tailing from table 1 the combined products on the 20 and the 35 mesh screens contained 86.3 per cent of the total zinc in the tailing, and a much lower recovery would have resulted except for the high zinc tenor of the feed. Because of this high zinc content, and with fairly good classification, table 1 showed a zinc recovery of 87.5 per cent.

The classification of the feed to the second table was reasonably good in view of the character of the ore. The relatively high zinc content in the coarsest screen product of the tailing, representing 34.3 per cent of the total loss from the table, shows the presence of chatty material, and indicates, as in table 1, the necessity of regrinding middlings from tables treating such material. The length of stroke was too long for the size of material treated. In view of the low zinc content of the feed and the character of the ore the percentage of recovery was fairly good.

The low percentage of recovery with the third table resulted from poor classification, as shown by the proportion of slimes, 26.4 per

cent, in the feed to this table. Most of the loss in the tailing was in the material passing the 200-mesh screen, which represented 50.3 per cent of the total zinc in the tailing. The length of stroke was too long for the size of material treated.

As the feed to the fourth table assayed 1.80 per cent zinc, a relatively high recovery was not possible. The classification of the feed was not of the best. If the classification of the feed to table 3 had been such that a considerable part of the material passing the 200-mesh screen had gone to table 4, the writer believes that better work would have been done by both tables. Still better results would have followed treating the slimes on slime tables with a proper thickness of pulp. Also, the length of stroke of this table was too long and the speed too slow for the size of material treated.

COMPARISON OF RESULTS WITH THE FOUR TABLES.

The chief points brought out in the results just described are (1) the necessity of regrinding middlings, (2) the necessity of good classification of feeds, and (3) the use of speeds and lengths of stroke best suited to the material treated. As these points are essential to good table practice, more attention should be paid to them by the operators or mill foremen.

The total percentage of zinc recovery made from the four tables, 74.3, was fairly good considering the character of the ore. The concentrates produced contained only 42.0 per cent zinc, but were retreated over separate tables later.

Although the relatively poor work of each table was due mostly to improper classification, wrong speeds and lengths of stroke, as shown by Table 27, contributed to the low recoveries.

The screen analysis of the total tailings from all four tables, presented in Table 28, shows that the screen products on the 20 and 35 mesh screens contained 1.59 and 1.90 per cent zinc, respectively, and that the combined zinc content of these two products represented 53.25 per cent of the total zinc contained in the tailings, and 39.0 per cent of the total material discarded as tailings.

TABLE 28.—Results of analyses of total tailings from all four tables.

Size of screen, mesh.	Size of openings.		Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
	Mm.	Inches.			
10	1.651	0.065			
20	.833	.0328	10.2	1.59	11.15
35	.417	.0164	28.8	1.90	42.1
65	.208	.0082	18.2	.80	11.15
100	.147	.0058	12.3	.53	5.15
150	.104	.0041	9.6	.50	4.3
200	.074	.0029	5.6	.80	3.85
<200	.074	.0029	15.3	1.90	22.30
Total or average.....			100.0	1.30	100.00

If the table section of the mill is assumed to have produced 100 tons of tailings a day, the two screen products on 20 and 35 mesh screens would have yielded 39 tons of tailings assaying 1.82 per cent zinc, the calculated assay value of the two combined products. As this 1.82 per cent represents metallic zinc, it would figure 3.64 per cent in terms of 50 per cent zinc concentrates. The quantity of 50 per cent zinc concentrates contained in the 39 tons of tailings would be 1.419 tons. If this material had been reground and treated over separate tables, a 60 per cent recovery should have been possible, or a saving of 0.851 ton of concentrate. If only 25 tons of tailings was produced from the tables there would have been a saving of 0.213 ton from these 25 tons, and if 50 tons had been treated, a saving of 0.425 ton. This simple example, one of several that might be figured out, shows that a few changes in the flow sheet of a mill may pay for themselves within a short time.

TESTS WITH 10 TABLES TREATING FAIRLY "FREE MILLING" ORE.

In Table 29 are the results obtained from the table section of a mill treating an ore of a fairly free character over 10 concentrating tables. The overflow from the last classifier, consisting mostly of slimes, was returned to the settling tanks. Since the tests were made, however, one Dorr thickener and four additional tables have been added to take care of the slimes overflowing from the settling tanks and classifiers in the mill.

TABLE 29.—Results of tests with 10 concentrating tables treating rather free milling ore.

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailings.		
		Millimeters.	Inches.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
Table 1 (speed 239; 1-inch stroke).	10	1.651	0.065	0.40	0.98	0.10			
	20	.833	.0328	.87	.72	.15	0.58	0.41	0.4
	25	.417	.0164	44.18	1.08	9.90	82.85	.56	15.6
	35	.208	.0082	35.06	4.86	35.25	33.68	.52	19.95
	100	.147	.0058	10.47	13.87	30.05	4.37	1.75	8.7
	150	.104	.0041	1.41	16.68	4.9	.85	2.22	2.15
	200	.074	.0029	8.13	12.90	13.7	.92	3.30	3.45
	<200	.074	.0029	2.48	11.58	5.95	6.75	6.49	49.85
Total or average..				100.00	4.83	100.00	100.00	.58	100.00
Table 2 (speed, 242; 1-inch stroke).	35	0.417	0.0164	36.55	0.75	5.95	45.20	0.35	23.80
	65	.208	.0082	45.61	3.30	22.8	43.14	.55	35.70
	100	.147	.0058	10.79	16.30	38.3	7.49	2.05	23.10
	150	.104	.0041	1.43	20.80	6.35	1.83	2.10	5.78
	200	.074	.0029	3.54	14.80	11.45	.89	2.40	3.24
	<200	.074	.0029	2.08	11.40	5.15	1.45	3.85	8.38
Total or average..				100.00	4.70	100.00	100.00	10.66	100.00

TABLE 29.—Results of tests with 10 concentrating tables treating rather free milling ore—Continued.

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Milli-meters.	Inches.	Percent-age of weight.	Zinc assay, per cent.	Percent-age of total zinc content.	Percent-age of weight.	Zinc assay, per cent.	Percent-age of total zinc content.
Table 3 (speed, 242; 1½-inch stroke).	35	0.417	0.0164	15.73	0.51	1.6	25.98	0.26	8.6
	65	.208	.0082	47.87	1.44	13.8	56.26	.72	51.7
	100	.147	.0058	20.53	10.80	44.3	12.07	1.60	24.7
	150	.104	.0041	3.96	12.50	9.9	.75	1.81	1.7
	200	.074	.0029	7.95	13.88	21.25	3.47	1.64	7.3
	<200	.074	.0029	3.96	11.60	9.15	1.47	3.15	6.0
Total or average..				100.00	5.0	100.00	100.00	.78	100.0
Table 4 (speed, 243; 1-inch stroke).	35	0.417	0.0164	4.21	0.36	0.3	7.59	0.26	3.45
	65	.208	.0082	40.04	.98	8.2	58.82	.36	37.30
	100	.147	.0058	30.75	4.55	29.25	25.14	.83	36.70
	150	.104	.0041	11.69	11.10	27.1	4.89	1.34	11.55
	200	.074	.0029	7.33	12.59	19.3	1.97	1.34	4.65
	<200	.074	.0029	5.98	12.70	15.85	1.59	2.27	6.35
Total or average..				100.00	4.78	100.00	100.00	0.57	100.00
Table 5 (speed, 245; 1-inch stroke).	35	0.417	0.0164	1.00	0.74	0.15	1.39	0.45	0.60
	65	.208	.0082	23.89	.52	2.50	34.51	.40	13.40
	100	.147	.0058	39.99	3.62	26.00	42.20	.80	32.70
	150	.104	.0041	15.46	6.40	18.8	13.46	1.00	13.1
	200	.074	.0029	11.38	10.31	23.45	5.02	1.25	6.1
	<200	.074	.0029	8.30	15.10	25.1	3.42	2.50	34.1
Total or average..				100.00	5.0	100.00	100.00	1.03	100.00
Table 6 (speed, 245; 1-inch stroke).	65	0.208	0.0082	5.04	0.55	0.53	18.61	0.47	5.8
	100	.147	.0058	27.44	1.10	5.79	62.42	1.32	55.0
	150	.104	.0041	28.78	3.70	20.43	6.28	2.58	10.80
	200	.074	.0029	21.72	7.70	32.10	6.16	2.88	11.85
	<200	.074	.0029	17.02	12.60	41.15	6.55	3.78	16.55
Total or average..				100.00	5.22	100.00	100.00	1.50	100.00
Table 7 (speed, 270; 1½-inch stroke).	35	0.147	0.0164	0.16	1.60	0.06	0.45	0.72	0.25
	65	.208	.0082	2.58	1.38	.70	5.02	.51	2.25
	100	.147	.0058	43.82	2.45	21.40	40.51	.70	24.45
	150	.104	.0041	20.57	4.25	17.42	27.48	1.02	24.45
	200	.074	.0029	19.67	7.30	28.64	14.26	1.55	19.10
	<200	.074	.0029	13.18	12.10	31.78	12.28	2.78	29.5
Total or average..				100.00	5.01	100.00	100.00	1.16	100.00
Table 8 (speed, 270; 1½-inch stroke).	35	0.417	0.0164	0.45	3.85	0.35	1.37	1.15	1.15
	65	.208	.0082	1.26	3.13	.80	.78	.40	.20
	100	.147	.0058	29.44	1.22	7.1	14.35	.40	4.20
	150	.104	.0041	35.21	3.76	26.1	35.07	.90	23.05
	200	.074	.0029	18.89	6.69	24.9	29.53	1.45	31.90
	<200	.074	.0029	14.75	14.00	40.75	18.90	2.90	40.10
Total or average..				100.00	5.07	100.00	100.00	1.37	100.00
Table 9 (speed, 274; 1-inch stroke).	35	0.417	0.0164	1.04	4.70	0.94			
	65	.208	.0082	.92	2.80	.49			
	100	.147	.0058	21.68	3.35	14.05	8.24	0.50	2.45
	150	.104	.0041	40.73	4.30	33.90	32.59	.60	11.75
	200	.074	.0029	26.92	6.10	31.76	34.17	1.50	30.90
	<200	.074	.0029	8.71	11.20	18.86	25.00	3.64	54.90
Total or average..				100.00	5.17	100.00	100.00	1.66	100.00

TABLE 29.—Results of tests with 10 concentrating tables treating rather free milling ore—Continued.

Table No.	Size of screen, mesh.	Size of openings.		Feed.			Tailing.		
		Millimeters.	Inches.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.	Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc content.
Table 10 (speed, 276; 4-inch stroke).....	65	0.208	0.0082	0.70	5.99	0.75	1.86	1.73	1.7
	100	.147	.0058	3.78	1.24	.85	4.58	.51	1.25
	150	.104	.0041	27.08	1.44	7.0	13.01	.63	4.3
	200	.074	.0029	42.15	5.30	40.2	38.91	1.60	32.65
	<200	.074	.0029	26.29	10.82	51.2	41.64	2.75	60.1
Total or average..				100.00	5.55	100.00	100.00	1.90	100.00

SUMMARY.

Table No.	Speed, strokes per minute.	Length of stroke.	Zinc content of feed.	Zinc content of concentrate.	Zinc content of tailing.	Percentage of zinc recovery.
		Inches.	Per cent.	Per cent.	Per cent.	
Table 1.....	239	1	4.83	58.20	0.88	83.1
Table 2.....	242	1	4.70	61.80	.66	86.9
Table 3.....	242	1½	5.00	58.78	.78	85.8
Table 4.....	243	1½	4.78	53.92	.57	89.0
Table 5.....	245	1	5.00	58.00	1.03	80.5
Table 6.....	245	1½	5.22	54.05	1.50	73.3
Table 7.....	270	1½	5.01	55.44	1.16	78.5
Table 8.....	270	1½	5.07	45.16	1.37	75.0
Table 9.....	274	1½	5.17	49.60	1.66	70.2
Table 10.....	276	1½	5.55	50.82	1.90	68.3

GENERAL CONSIDERATIONS OF TABLE PRACTICE.

In several mills in the Joplin district the table practice is good; in most mills, however, attention is paid to the grade of concentrates produced rather than to the work of each table. If the essentials of good table practice were carried out, a higher recovery of products of the desired grades would be possible in many mills. Detail tests made from time to time would show what each table is doing. The following considerations bearing on higher recoveries by tables seem to be worth mentioning.

CAPACITY OF TABLES.

The capacity of a table depends on the rate at which the ore travels over the deck, and therefore chiefly on the length of stroke and the number of strokes per minute. In general, the capacity ranges from 4 to 40 tons in 24 hours, according to the fineness or coarseness of the feed; "roughing" tables usually have two or three times the capacity of ordinary sand tables treating the same kind of material.

The main bulk of the sand that reaches the tables is the undersize of the dewatered tailings from the rougher jigs and that in the overflow waters from the other jigs. A common method of dewatering the tailings from the rougher jigs is with a trommel placed at right

angles to the end of the jig. The tailings flow off an apron at the end of the jig and fall on the outside of the revolving screen. The undersize is washed through by the water in the tailings. A spray of water plays on the oversize before it leaves the trommel and forces more of the undersize through the screen openings, increasing somewhat the screen efficiency. These dewatering trommels are 4 to 5 feet in diameter, with screen openings of usually $1\frac{1}{2}$ to 2 mm., and revolve in the direction of the flow, at speeds of $1\frac{1}{2}$ to 4 revolutions per minute.

The efficiency of the dewatering trommel, or the proportion of the feed that passes the openings, is 40 to 60 per cent. This low efficiency results from the amount of undersize, depending almost entirely upon the amount of water used to force it through. Having determined the efficiency of the dewatering screen, and knowing approximately the tonnage passing over the rougher jig, one can estimate fairly closely the quantity of undersize from this screen that passes into the sand tanks.

In mills where dewatering screens are used, it would seem advisable to determine their efficiency, as this varies widely and a considerable proportion of the table-size material containing zinc may pass over the screen with the coarse tailings. If such be the case, increased screening efficiency and better saving of the values can readily be effected.

Screen analyses have shown that the tailings from rougher jigs before dewatering usually contain 10 to 30 per cent of table-size material, according to the character of the ore and the size to which it has been crushed before jigging.

The results obtained in treating classified material on nine concentrating tables in actual practice in a mill treating sheet-ground ore are shown in Table 30. The table shows the rate of feed, the percentage of zinc recovery, and other data, for each concentrating table.

TABLE 30.—*Results of tests showing tonnage of a classified feed in mill treating sheet-ground ore on nine concentrating tables.^a*

[Covering a period of 75 hours actual running.]

Table No.	Zinc content of—			Quantity of zinc concentrate produced.	Quantity of zinc in feed.	Quantity of zinc in concentrate.	Percentage of zinc recovery.	Quantity of tailings.	Total quantity of feed to tables.	Rate of feed, tons per hour.
	Feed.	Tailing.	Concentrate.							
	Per ct.	Per ct.	Per ct.	Tons.	Tons.	Tons.		Tons.	Tons.	
1.....	4.7	1.00	51.8	7.105	4.582	3.680	80.3	90.385	97.50	0.73
2.....	4.3	.60	51.7	7.713	4.584	3.988	87.0	98.887	106.60	.795
3.....	4.0	.75	53.1	5.481	3.582	2.910	82.4	84.059	89.54	.67
4.....	4.0	.70	53.7	3.654	1.990	1.662	83.5	46.106	49.76	.37
5.....	4.1	1.10	53.2	3.653	2.602	1.943	74.7	59.807	63.46	.47
6.....	4.0	1.10	51.3	1.827	1.265	.938	74.1	29.803	31.63	.235
7.....	4.8	2.35	47.9	1.015	.905	.486	53.7	17.835	18.85	.14
8.....	5.2	3.50	44.2	1.217	1.516	.538	35.5	27.933	29.15	.22
9.....	5.25	2.20	34.3	1.015	.560	.348	62.1	9.645	10.66	.08
	4.34	1.1	50.5	32.680	21.586	16.493	76.5	464.470	497.15	3.71

^a For the size of material treated on each table see Table 21 (p. 81).

PRODUCTS FROM THE TABLES.

As previously stated, the usual practice in the mills of the Joplin district is to make a clean lead concentrate and a clean zinc concentrate and to return the intermediate mixed lead, iron, and zinc concentrate to the original table feed. It is believed that a better practice would be to save this product and treat it separately from time to time, as under present practice the iron sulphide accumulates and must eventually contaminate the lead or zinc concentrates, unless eliminated in some way. If the lead content of the ore is small, the iron sulphide might better go with the lead concentrate rather than be returned to the feed.

Middlings from tables are usually returned to the original table feed, although in a few mills they are treated on separate tables. Returning these middlings to the feed cuts down the relative capacity of each table and lowers their efficiency, especially if the table middlings are not reground before being returned. When the middlings consist of free mineral and gangue, already a concentrated product, it would seem advisable to treat this product on separate tables. By treating the middlings from tables separately, the quantity of middlings produced from the primary tables is cut down, and the relative capacity of the tables increased.

STAGE CONCENTRATION.

Where the ratio of concentration is as high as it is in the mills of this district, the use of a "roughing" and "cleaning" system of concentration by tables would seem preferable in order to assure high recoveries. Such a change would require the use of two or more roughing tables of relatively large capacity and the treatment of the enriched product from these tables on the usual sand tables. A considerable part of the lead might possibly be removed by this treatment before the material reached the sand tables. The chief advantages of the plan would be the higher recoveries obtainable and the greater relative capacity of the sand tables, or possibly a decrease in the number of sand tables used. This plan is being followed in a few of the mills with good results.

Another effective practice in treating fine material on tables is to use, for each classification, two tables, one placed a table's width to the side of the other and about 2 feet lower, both being set on a concrete foundation. From the first and higher table, concentrates only are taken. The mixed product of lead, iron, and zinc from the concentrate end, the middling, and the tailing, from the first table all go by gravity to the feed box of the second table. From this second table concentrates are produced, the middling is re-

ground and sent over separate tables, and the tailing is discarded. The writer had the privilege of testing this system of tabling lead and zinc ores at a mill in Sardinia, Italy, by which high recoveries were obtained.

RETREATMENT OF CONCENTRATES ON TABLES.

When the concentrates from the jig and table sections treating the coarse material are fairly low grade, containing 2 to 3 per cent lead, 4 to 10 per cent iron, and 40 to 50 per cent zinc, the remainder being gangue, it has been found that crushing the material to table size and treating it on tables yields a much better grade of product.

In mills where the concentrates have been re-treated in this way the lead content has been reduced to 0.2 to 0.4 per cent and the iron content to less than 2 per cent. However, in one mill the iron content was reduced from 8 or 9 to only 5 per cent, while the zinc tenor was increased several points. When the content of iron sulphide is high, elimination of that mineral from the zinc-blende concentrates is difficult because of the small difference in specific gravity between the two minerals, whereas the lead content is fairly easy to cut down. To obtain good results with minerals differing but slightly in specific gravity, very close classification is necessary. Therefore, classification by sizing, also known as grading, would seem feasible. The concentrates could be screened to three or more different sizes and the sized products treated on separate tables, the concentration then depending more on the differences in specific gravity of the minerals. The flow sheet for such retreatment on tables would have to be designed with reference to the quantity of concentrates to be treated and the proportionate quantity of the various screen products.

If the concentrates produced from the mill, before retreatment should be high in iron and consequently low in zinc, say about 40 per cent zinc, and the ore from the mine should be fairly uniform, it would probably pay to install a small roasting and magnetic separating plant, rather than to treat the ore on tables. Giving the concentrates a superficial roast and then passing them through magnetic separators has proved satisfactory in the Wisconsin field.^a

ESSENTIAL CONDITIONS FOR GOOD TABLE PRACTICE.

The essential points to be considered in good table practice can be summed up as follows: (1) The character of the ore, (2) the richness of the feed, (3) the size of the material treated, (4) the quantity of feed, (5) the previous classification of the feed, (6) the uniformity of the feed coming onto the table, (7) the thickness of the pulp treated,

^a Wright, C. A., Mining and milling of lead and zinc ores in the Wisconsin district, Wis., Tech. Paper 95, Bureau of Mines, 1915, pp. 27-28.

(8) the operating conditions of the table, such as speed, length of stroke, amount of wash water, slope of deck, and firmness of foundations, (9) the character and quantity of the middlings produced and their further treatment, (10) the kind and grade of the concentrates produced, (11) the further treatment of the "returns" or mixed concentrate products of lead, iron, and zinc, (12) the possibilities of stage concentration by tables.

All of these conditions are, of course, dependent on the quantity of ore to be treated, the number of tables available or that can be added, and the personal attention given to the tables by the men in charge.

CHARACTER OF OVERFLOW FROM SETTLING TANKS.

Anyone familiar with the mills in the Joplin district knows that the practice has been to produce the smallest amount of fine material possible, as a large proportion of the losses has been in the finest sizes. The coarse tailings from the "rougher" jigs, after being dewatered, are seldom reground for further treatment, but are sent direct to the tailings pile. Consequently, the quantity of slimes is relatively small as compared with that produced by jigs and tables in other parts of the country.

As the treatment of slimes containing zinc blende is now possible by improved concentrating methods, especially by flotation, the production of a somewhat larger proportion of fine material should not be considered such a disadvantage in the milling of the Joplin ores.

LOSSES IN OVERFLOW.

In figuring the percentage of recovery in a mill all of the losses should be included, a part of which is in the overflow waters from the settling tanks. The zinc assays, as a rule, are relatively much higher in the slimes and overflow from the settling tanks than in the table sands and coarser material.

RESULTS OF MILL TEST IN MILL TREATING SHEET-GROUND ORE.

The results of a mill test showing the quantity of zinc lost in the overflow, the relatively higher assays in the finer sizes of the mill tailings, not including the overflow, and the percentage of loss in the various screen products, are presented in Table 31.

TABLE 31.—Results of a mill test in mill treating sheet-ground ore.

Screen analysis of mill tailings not including overflow.					Zinc (Zn) content.		
Size of mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
	Mm.	Inches.					
	13.33	0.525	<i>Per cent.</i> 6.89	<i>Per cent.</i> 33.97	<i>Per cent.</i> 0.30	5.6	-----
3	6.690	.263	32.08	38.97	.22	19.3	24.9
6	3.327	.131	26.75	65.72	.20	14.6	39.5
10	1.651	.065	18.40	84.12	.19	9.5	49.0
20	.833	.0323	9.07	93.19	.30	7.4	56.4
35	.417	.0164	2.70	95.89	.56	4.1	60.5
65	.208	.0082	1.80	97.69	1.27	6.3	66.8
100	.147	.0058	.44	98.13	2.93	8.5	70.3
150	.104	.0041	.39	98.52	3.35	8.6	73.9
200	.074	.0029	.21	98.73	5.70	8.3	77.2
<200	.074	.0029	1.27	100.00	6.55	22.8	100.00
Total or average.....			100.00	-----	.37	100.00	-----
Check sample.....			-----	-----	.41	-----	-----

Ore treated.....	Tons.	1,994
Concentrates produced.....		35.5
Tailings.....		1,958.5
Quantity of solids in overflow from settling tanks.....		34.73
Quantity of tailings, not including material in overflow.....		1,923.77
Quantity of metallic zinc in—	Per cent.	
Concentrates.....	59.4	21.087
Tailings without overflow.....	.41	7.887
Overflow.....	5.01	1.740
Total quantity of zinc in ore treated.....		30.714
Percentage of zinc recovery $(27.087+30.714) \times 100$	68.61	

The results of this test, made in a mill treating "sheet-ground" ore, show a zinc recovery of 68.6 per cent.

As shown by the screen analysis and the zinc assays of the various screen products in the mill tailings, not including the overflow from the settling tanks, 22.8 per cent of the total zinc was in the material finer than 200 mesh. This material represented 1.27 per cent of the total weight of the mill tailings. The quantity of solids in the overflow for the quantity of ore treated (1,994 tons) was 34.73 tons, with an assay value of 5.01 per cent zinc. The quantity of zinc in the overflow, therefore, was 1.740 tons. As the mill tailings contained 7.887 tons of zinc, the total quantity of zinc lost for the tonnage treated was 9.627 tons. From these figures, the percentage of the total loss represented by the overflow was $(1.740 \div 9.627) \times 100 = 18.1$ per cent. If the percentage of recovery made by the mill not including the values lost in the overflow from the settling tanks were calculated, the result would show a zinc recovery of 72.8 per cent. In other words, the apparent recovery would be 4.2 per cent better if the values in the overflow were not included in the total zinc lost.

RESULTS OF SCREEN ANALYSIS OF SOLIDS IN OVERFLOW FROM SETTLING TANKS AT MILL TREATING SHEET-GROUND ORE.

To show more fully the values in the various sizes of solids in the overflow from settling tanks at a representative mill treating sheet-ground ore, the writer presents the following screen analyses and the zinc assays of the different screen products.

TABLE 32.—*Screen analysis of solids in overflow from settling tank of mill treating sheet-ground ore.*

Size of screen mesh.	Size of openings.		Weight of screen products.	Cumulative weight.	Zinc content.		
	Mm.	Inches.			By assay.	Percentage of total zinc.	Cumulative percentage of total zinc.
100	0.147	0.0068	Per cent. 8.19	-----	Per cent. 5.00	6.05	-----
150	.104	.0041	7.60	15.79	4.50	5.06	11.11
200	.074	.0029	8.22	24.01	5.20	6.32	17.43
<200	.074	.0029	75.99	100.00	7.35	82.57	100.00
Total average.....			100.00	-----	6.76	100.00	-----
Check sample.....			-----	-----	6.60	-----	-----

RESULTS OF TESTS OF OVERFLOW FROM SETTLING TANKS AT SEVERAL MILLS.

Table 33 shows the weight of solids per gallon and the lead and zinc assays of solids in the overflow from the settling tanks of several mills.

TABLE 33.—*Data on overflow from settling tanks at seven different mills.*

Mill No.	Quantity of solids. per gallon. ^a	Ratio of water to solids.	Per cent of solids in overflow.	Assay of solids.		Remarks.
				Lead (Pb).	Zinc (Zn).	
	<i>Grams.</i>			<i>Per cent.</i>	<i>Per cent.</i>	
1	11.67	323:1	0.309	1.20	5.01	Single tanks.
2	12.67	298:1	.335	2.41	7.04	Do.
3	13.85	272:1	.365	1.45	5.25	Do.
4	9.35	405:1	.246	.37	6.38	Double tanks.
5	25.33	161:1	.619	.14	1.13	Single tanks.
6	9.66	391:1	.254	2.78	6.60	Double tanks.
7	9.30	406:1	.245	1.90	8.40	Do.

^a 1 gallon contains 3,785 c. c., or 3,785 grams of water (specific gravity, 1.00); specific gravity of solids was taken as 2.7.

METHOD OF CALCULATING VALUE OF LOSSES AND PROBABLE RECOVERY.

As the table shows, the percentage of solids in the overflow waters was low. In order to treat the contained slimes these overflows would have to be thickened to a fairly thick pulp. In figuring the

zinc loss in the overflow, the number of gallons of overflow a minute would have to be measured at each individual mill. The overflow from the settling tanks is about 700 to 1,200 gallons a minute, depending, of course, on the tonnage treated and the quantity of water used. If 1,000 gallons a minute be taken as an average, an approximate figure for the zinc values lost during a period of 10 hours can be determined for the mills represented in the preceding table, as shown in Table 33 following.

TABLE 34.—*Estimated zinc losses in overflow at seven mills, calculated from data in Table 33.*

[Zinc values figured on the basis of 1,000 gallons per minute overflow.]

Mill No.	Quantity of solids in overflow for 10 hours.		Zinc content.	Quantity of zinc in overflow for 10 hours.	Quantity of "50 per cent zinc" concentrates in overflow for 10 hours.
	Kilograms.	Pounds. ^a			
1	7,002	15,437	<i>Per cent.</i>	<i>Pounds.</i>	<i>Pounds.</i>
2	7,602	16,766	5.01	773	1,546
3	8,310	18,320	7.04	1,159	2,318
4	5,698	12,341	5.25	902	1,804
5	13,998	30,860	6.38	787	1,574
6	5,796	12,778	1.13	249	498
7	5,580	12,301	6.60	843	1,686
			8.40	1,033	2,066

^a 1 pound is figured as equal to 453.6 grams.

After computing the quantities of zinc lost in the overflow for 10 hours, and figuring that a saving of 50 to 80 per cent is possible by flotation or some other method, values can be calculated in dollars and cents. Assuming that the product obtained, after treatment, will have a value of \$30, \$40, \$50, or \$60 per ton for 50 per cent zinc concentrates, and figuring these values on a 50 and an 80 per cent zinc recovery, one obtains the figures as shown in Table 35.

TABLE 35.—*Value of "50 per cent" zinc concentrates in overflow during 10 hours from various mills.*

Mill No.	Quantity of "50 per cent zinc" concentrates in 10 hours. ^a	Value at \$30 per ton.	Value at \$40 per ton.	Value at \$50 per ton.	Value at \$60 per ton.
	<i>Pounds.</i>				
1	1,546	\$23.19	\$30.92	\$39.65	\$46.38
2	2,318	34.77	46.36	57.95	69.54
3	1,924	28.96	38.48	48.10	57.72
4	1,574	23.61	31.48	39.35	47.22
5	698	10.47	13.90	17.45	20.94
6	1,698	25.19	33.72	42.15	50.58
7	2,066	30.99	41.32	51.65	61.98

^a Figures are those given in Table 34.

TABLE 35.—Value of "50 per cent" zinc concentrates in overflow during 10 hours from various mills—Continued.

VALUES BASED ON 50 PER CENT SAVING.

Mill No.	Quantity of "50 per cent zinc" concentrates in 10 hours, pounds.	Value at \$30 per ton.	Value at \$40 per ton.	Value at \$50 per ton.	Value at \$60 per ton.
1	1,546	\$11.59	\$15.46	\$19.82	\$23.19
2	2,318	17.38	23.18	28.97	34.77
3	1,924	14.48	19.24	24.05	28.96
4	1,574	11.80	15.74	19.67	23.61
5	698	5.23	6.98	8.72	10.47
6	1,686	12.59	16.86	21.07	25.19
7	2,066	15.49	20.66	25.82	30.99

VALUES BASED ON 80 PER CENT SAVING.

1	1,546	\$18.55	\$24.74	\$31.72	\$37.10
2	2,318	27.82	36.99	46.36	55.63
3	1,924	23.17	30.78	38.48	46.18
4	1,574	18.99	25.18	31.48	37.78
5	698	8.38	11.17	13.96	16.75
6	1,686	20.15	26.98	33.72	40.48
7	2,066	24.79	33.06	41.32	49.58

At No. 1 mill there was 1,546 pounds of 50 per cent zinc concentrates in the overflow during 10 hours. This quantity, figured at \$30, \$40, \$50, and \$60 per ton, gives values of \$23.19, \$30.92, \$39.65, and \$46.38. If a 50 per cent saving be assumed, the values become \$11.59, \$15.46, \$19.82, and \$23.19. The net profit will be the difference between these values and the cost of treatment. The figures given are based on a 10-hour day; if the mills were in use 20 hours or 24 hours a day the values would be proportionately greater.

These figures bring out the possibilities of saving more of the minerals now going to waste and show one of the reasons why 30 to 40 per cent of the zinc in the ore treated in the mills of the Joplin district is lost. Hence the management of each mill should determine what percentage of the total loss from the mill is represented by the overflow from the settling tanks, and, if the loss warrants them, what improvements or changes will make a greater commercial saving. The commercial possibilities for a greater saving of the values contained in the slimes or overflow slimes under the present milling practice will, of course, depend on the quantity of fine material produced, so that, unless a slime-treatment unit or a flotation unit of small enough capacity can be added, any saving on a profitable basis will probably be limited to the larger mills of the district.

However, as has previously been stated, the fact that it is now possible to obtain a relatively high recovery by flotation from slimes containing sulphides of lead and zinc indicates that a greater proportion of the fine material produced by finer grinding could be

profitably saved with proper equipment. Especially the middlings from tables and the "chats," or particles containing included mineral, from the jigs, could be ground finer.

POSSIBILITIES OF FLOTATION.

A number of flotation tests of the lead and zinc ores of this district have shown that floating the sulphides is relatively easy. The ore consists mainly of sphalerite, the zinc sulphide, with varying smaller quantities of lead sulphide, galena, and flint being the chief gangue material. Certain problems, however, such as the quantity of material that can be floated without further crushing and the possibilities of finer grinding, must be worked out to insure success on a commercial scale. Therefore, the mill men of this district, because of the difficulties in applying flotation to the treatment of these ores, have on the whole been somewhat reluctant to adopt the process. Although at present flotation does not hold as important a place in the treatment of Joplin ores as in many of the larger copper and zinc mills of the West, yet the writer believes that before long many of the mills will have small flotation units for saving a large proportion of the zinc now going to waste in the fine material. Five or six mills are using flotation successfully, and others are testing it. The grade of flotation concentrates produced by the mills using the flotation process varies from about 55 to 59 per cent zinc.

In the Joplin district the mills are small, their capacity being 100 to 500 tons of ore treated in 10 hours. They are equipped with crusher, rolls, jigs, and tables. At most of them the material that reaches the jigs has passed a $\frac{1}{4}$ -inch trommel screen. The middlings or "chats" from the jigs are recrushed for further treatment on sand jigs or tables, or, as is the practice in some mills, returned to the rougher jigs. Middlings from the tables are seldom reground. The first point to be considered, therefore, is how much material now being produced is fine enough to be amenable to flotation, or in other words, what percentage of the total tonnage treated is crushed to a suitable size for flotation and what proportion of this size can be saved for retreatment.

PROPORTION OF MATERIAL OF SUITABLE SIZE FOR FLOTATION.

RESULTS OF TEST.

To show more clearly the quantity of material available and of suitable size for flotation the results of a simple mill test are presented in Table 36 following. The ore treated was from a hard-ground mine, better known as sheet-ground ore, consisting chiefly of flint as the gangue with a small percentage of zinc and practically no lead.

TABLE 36.—Results of sizing test of tailings in a mill treating sheet-ground ore.

Screen analysis of mill tailings not including overflow.			Weight of screen products.	Cumula- tive weight.	Zinc content.		
Size of mesh.	Size of openings.				By assay.	Percent- age of total zinc.	Cumula- tive per- centage of total zinc.
	Mm.	Inches.					
	13.33	0.525	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>		
-----			6.89		0.30	5.6	
3	6.680	.263	32.08	38.97	.22	19.3	
6	3.327	.131	26.75	65.72	.20	14.6	
10	1.651	.065	18.40	84.12	.19	9.5	
20	.833	.0328	9.07	93.19	.30	7.4	
35	.417	.0164	2.70	95.89	.56	4.1	
65	.208	.0082	1.80	97.69	1.27	6.3	
100	.147	.0058	.44	98.13	2.93	3.6	
150	.104	.0041	.39	98.52	3.35	3.3	
200	.074	.0029	.21	98.73	5.70	3.3	
<200	.074	.0029	1.27	100.00	6.55	22.8	
Total.....			100.00	-----	a .37	100.0	
Check sample.....				-----	.41	-----	

a Calculated.

		Tons.
Ore treated.....		1,994
Concentrates produced		35.5
Tailings.....		1,958.50
Solids in overflow from settling tanks.....		34.73
Tailings, not including solids in overflow.....		1,923.77
Metallic zinc in—	Per cent.	
Concentrates.....	59.40	21.087
Tailings without overflow.....	.41	7.887
Overflow.....	5.01	1.740
Total quantity of zinc in ore treated.....		30.714
Percentage of zinc recovery $(21.087+30.714) \times 100$	68.6	

DISCUSSION OF RESULTS OF TEST.

The results of the tests in the preceding table indicate a zinc recovery of 68.6 per cent. As shown by the screen analysis of the mill tailings, not including the solids in the overflow from the settling tanks, 2.31 per cent passed the 65-mesh screen, a product fine enough for flotation. The quantity treated during this test was 1,994 tons, containing 44.44 tons of material passing the 65-mesh screen. The calculated zinc content of this size material was 5.24 per cent, or 2.328 tons. On the assumption that the overflow is suitable to flotation, there are 34.73 tons of solids containing 5.01 per cent zinc, or 1.74 tons of zinc. The sum of these two products makes a total of 4.068 tons of zinc in the material suitable to flotation, or 79.17 tons with a calculated content of 5.14 per cent zinc. If this material would be available and if 80 per cent of its zinc content could be recovered by flotation, 3.256 tons of zinc would be saved. The total zinc content of the 1,994 tons of ore treated was 30.714 tons. Adding to the 21.087

tons of zinc saved in the concentrate the 3.256 tons that could be recovered from the 79.17 tons of floatable material by flotation, we have 24.343 tons of zinc saved out of a possible 30.714 tons, or a zinc recovery of 79.2 per cent. In other words, the recovery would be increased from 68.6 to 79.2 per cent, representing an additional saving of 10.6 per cent of the total zinc in the ore.

The above figures bring out the possibilities of marked savings by flotation with a relatively large tonnage. Unfortunately, the average daily tonnage treated by the mills is considerably less than 1,994 tons, the tonnage treated during the time the test was being made. It is reasonable to believe, however, that the mills treating 500 to 1,000 tons of ore daily could install a small flotation unit at a profit, the size of the unit depending on the quantity of fine material produced.

In milling soft-ground ore generally the proportion of fines is much larger than with hard-ground ore; hence enough fine material would be produced in a mill treating 200 to 300 tons a day to operate a 20 to 30 ton flotation unit.

DIFFICULTY OF GRINDING TO SIZES SUITABLE FOR FLOTATION.

The next step in determining the possibilities of flotation for the ores of the Joplin district is the profitable limit of fine grinding, which would be one of the main problems in flotation, as the gangue consists mostly of flint. The question, therefore, is one of fine grinding rather than of floating the mineral, at least in mills where not enough material is already available for profitable flotation. Many kinds of fine grinding mills have been tried in this district with more or less discouraging results, as the flint is highly abrasive. In fact, the fine grinding of Joplin ores, either by ball mills or some other means, is still an unsolved problem.

Certain of the mill products, however, would be less resistant and more easily ground. Some of these products are: (1) "chats" from the jigs, (2) tailings from the tables treating fine material, (3) all table middlings that need to be reground before further treatment, and (4) possibly the oversize often discarded from sand screens. The important factors to be considered in the possible treatment of these products by flotation are the cost of fine grinding, the quantity of material to be reground, and the zinc content of the material.

If the material contained 1 per cent metallic zinc, equivalent to 2 per cent of a 50 per cent zinc concentrate, and the concentrate was worth \$50 a ton, its value would be \$1 a ton or \$0.80 a ton on a basis of 80 per cent zinc recovery. For material containing 2 per cent zinc these values would be \$2 a ton and \$1.60 for 80 per cent recovery. In other words, it is possible to figure from the zinc content and the possible zinc recovery what value can be placed on the material to be reground and treated.

The cost of fine grinding would depend upon the type of grinding mills used and on the character and coarseness of the material. The greater the tonnage treated the less would be the unit cost of grinding and subsequent treatment. The possibilities are, therefore, greatly increased when one considers the larger tonnage available for flotation by finer grinding. In addition slimes or fine sands that are at present being treated on slime tables could be treated by flotation, and a higher zinc recovery effected than is usually possible on slime tables.

In view of the demand for high-grade zinc concentrates by smelters, zinc ores containing a comparatively large proportion of lead and iron sulphides would require either differential flotation or retreatment of the concentrates on tables. At a few mills the floatable material is given a preliminary treatment on tables to eliminate the lead and iron, but this requires extra tables and is of doubtful efficacy where the content of iron and lead in the feed is high.

Retreatment of the concentrates on tables seems to be the most feasible method of separating the lead and iron sulphides from the zinc blende and could be cheaply done. A small diaphragm pump, operated in closed circuit, could be used to return the tailings from the tables to the flotation system. This method of treatment has given satisfactory results in some of the local mills. The froth from the concentrates is largely "killed" by spraying with water, or by some other effective means, before it reaches the table. The ease with which the froth is killed depends largely upon the oil mixture or frothing agent used.

EXPERIMENTS WITH DIFFERENT OILS.

The composition of the gangue of the Joplin ores varies somewhat as does the mineral content of the ores. Consequently, although many oils are available that give acceptable results, there is no one oil or certain quantity of oil that under the same operating conditions will give the best possible results for all the lead and zinc ore of this district. Hence the best oils or the best mixture to use must be determined by experiment.

The Bureau of Mines, in cooperation with the Missouri geological survey, started flotation experiments in the Joplin district in the latter part of the year 1914 and continued them until the spring of 1915. In the first experiments different oils were tried in varying quantities to decide which oil was most suitable. When this point had been determined, similar tests were made with the addition of sulphuric acid in varying quantities. The effect of temperature was then considered.

The results of these tests indicated that separation of the sulphides from the gangue in ores of this district by flotation is not difficult, and that

a fairly good grade of concentrate can be obtained by the use of rougher and cleaner cells. The use of oils with a coal-tar base gave a high recovery, but a low grade of concentrate, a considerable amount of gangue being entrained in the froth. Other oils gave a higher grade of concentrate, but not as good a recovery under like conditions. The best results were obtained with wood creosote and pine oils. As coal tar and some petroleum products are good collectors and pine oil and other suitable oils from wood distillation are good frothing agents, a mixture of 80 per cent coal tar and 20 per cent pine oil or wood creosote, or some other combination of these oils, would seem to be best suited to the treatment of the Joplin ores by flotation.

Also, several tests were made with mixtures of turpentine and rosin. The rosin was dissolved in the turpentine in proportions of 10, 20, and 30 per cent, respectively. Most of the tests were roughing tests to determine what grades of concentrate and what percentages of recovery could be made with these mixtures. The 20 and 30 per cent mixtures gave the best results, both as to grade and recovery. When 1 to 5 pounds of mixture was used per ton of ore treated in an acid pulp of about 5:1 thickness (water to solids), the rough concentrates varied from 37 to 56 per cent in zinc content, and zinc recoveries from 50 to 80 per cent. In most tests the higher the grade of concentrate obtained the lower was the recovery. However, the relatively low recoveries may be considered good in view of the low zinc content of the heads, which assayed 1.05 to 2.18 per cent zinc. It is believed that with richer feeds much higher recoveries could have been obtained.

TESTS AT SALT LAKE CITY STATION.

To verify the data obtained from this preliminary work, slimes produced in a mill treating ore from one of the sheet-ground mines near Webb City, Mo., were shipped to the experiment station of the Bureau of Mines at Salt Lake City, Utah, where the bureau, in co-operation with the University of Utah, has been conducting flotation experiments. The results of the experiments, which were performed by O. C. Ralston and G. L. Allen, are presented in Tables 37 and 38, following. They show that the sphalerite can be floated from the gangue with comparative ease by using warm solutions and about 1 pound of any suitable oil, either from wood or coal distillation, per ton of ore, and that acidifying the pulp, although it does not seem to be necessary, permits the froth and tailing to separate more quickly. Cold solutions gave as high recoveries as warm solutions, but the grade of product was not as good.

TABLE 37.—*Results of roughing tests in flotation of slimes from sheet-ground ore.*^a

[Conditions of tests: Assay of slimes tested, 8.54 per cent zinc, 0.51 per cent lead, 0.7 per cent iron; charge used in each test, 500 grams, with 1,500 to 2,500 c. c. of water; temperature of pulp, 60° C., except in test 9, where it was 20° C.; length of agitation, 18 minutes.]

VARYING ACID CONTENT.

Test No.	Acid used, pounds per ton of ore treated.	Oil used.		Weight of froth, grams.	Grade of concentrate, per cent zinc.	Percentage of zinc recovery.	Remarks.
		Kind.	Quantity, pounds per ton of ore treated.				
1	7.36	Pine oil.....	1.37	105.5	35.9	88.8	Distinct froth line.
2	14.72	do.....	1.37	96.5	39.3	88.8	
3	36.8	do.....	1.37	95.5	38.9	87.0	
4	73.6	do.....	1.37	100.0	38.8	91.0	

VARYING OIL CONTENT.

Test No.	Oil used.		Weight of froth, grams.	Grade of concentrate, per cent zinc.	Percentage of zinc extraction.	Remarks.
	Kind.	Quantity, pounds per ton.				
5	Pine oil.....	0.5	104.5	38.5	89.5	Cold solution used (20° C.). Rapid separation. Very rapid separation. Strong froth. Strong froth, rapid separation. Strong froth.
6	do.....	1.0	86.5	43.2	87.5	
7	do.....	2.0	103.5	36.2	87.8	
8	do.....	4.0	94.0	39.7	87.5	
9	do.....	1.0	120.5	31.6	89.2	
10	Crude cedar oil.....	2.5	93.5	40.1	87.3	
11	Special pine-tar oil.....	1.8	73.5	49.6	85.5	
12	Wood creosote.....	1.7	93.0	40.8	89.0	
13	Coal creosote.....	2.0	78.0	46.8	85.8	
14	Eucalyptus oil.....	1.7	110.0	34.2	88.2	

Samples of each froth from the roughing tests were taken, and the rest was saved for cleaning tests. All of the roughing froths combined made enough material for two cleaning tests, the results of which were as follows:

TABLE 38.—*Results of cleaning tests with froth from roughing tests.*^a

TEST 1.

Quantity of froth used.....	grams..	500
Assay of head:		
Zinc content.....	per cent..	39.8
Lead content.....	do....	1.3
Quantity of product.....	grams..	366.5
Assay of product:		
Zinc content.....	per cent..	51.7
Lead content.....	do....	2.35
Percentage of recovery.....	do....	95.3

^a Tests by O. C. Ralston and G. L. Allen.

TEST 1.

Quantity of froth used.....	grams..	479
Assay of head:		
Zinc content.....	per cent..	39.8
Lead content.....	do....	1.3
Quantity of products:		
Froth.....	grams..	346
Middling.....	do....	47
Tailing.....	do....	77
Assay of products:		
Zinc content of—		
Froth.....	per cent..	51.5
Middling.....	do....	13.5
Tailing.....	do....	1.0
Lead content of—		
Froth.....	do....	2.04
Middling.....	do....	2.66
Tailing.....	do....	0.04
Percentage of recovery:		
Froth.....	do....	93.2
Middling.....	do....	3.3
Tailing.....	do....	4.0

DISCUSSION OF RESULTS OF TESTS.

Tests 1 to 4, Table 37, were made to determine the effect of different quantities of acid on the percentage of recovery, the grade of concentrate, and the sharpness of the line of separation between the yellow froth and white tailings, the quantity of oil used being constant. The results indicate that the quantity of acid used has little effect upon either the grade of concentrate or the percentage of recovery, but it was found that the line of separation between concentrate and tailing, as seen through the glass side of a laboratory machine, is much sharper and more quickly defined when a large quantity of acid is used.

The next four tests, Nos. 5 to 8, were run with varying amounts of oil, some acid having been added to permit the experimenters to see the froth and tailing separate more quickly and to clean the surfaces of the sulphide particles. No distinct effect was observed except that the use of 1 pound of oil per ton of material treated seemed to give the highest grade of concentrate. Test 9 was run to parallel test 6, except that a cold pulp at 20° C. was used in place of the warm pulp at 60° C. tests. The lower grade of the froth in test 9 indicates that the froth and tailing separate more readily at the higher temperatures. Tests 10 to 14 were run with various kinds of oils.

After the froth concentrates had been sampled, they were mixed together, and the two tests shown in Table 38 were run, the flotation cell being used as a "cleaner" unit. The previous tests had been run with the cell as a "rougher" unit, with a high recovery of zinc as the end in view. In cleaning the rougher froth a concentrate of high zinc tenor should be the aim, but these tests were carried to nearly com-

plete removal of the zinc. Hence only a small amount of middling was left for returning to the rougher cell, and the concentrates were perhaps not as high in zinc as could have been obtained by returning a larger quantity of middling.

CONCLUSIONS AS TO FLOTATION OF JOPLIN ORES.

The use of rougher and cleaner units of flotation cells seems essential in flotation of Joplin ores in order to obtain a high recovery and have the concentrates of acceptable grade.

In the flotation experiments made in the Joplin district, the thickness (ratio of solids to water) of the pulp treated ranged from 1 : 3 to 1 : 7. In practice the most favorable ratios for different materials would have to be determined by experiment. Generally a mixture of fine material and slimes requires a denser pulp, whereas for treating a mixture consisting wholly of slimes (finer than 200-mesh) the use of a thinner pulp is desirable. Thickening of the slimes as produced in the mills of this region would be necessary. This could be accomplished by the use of Dorr thickeners or some other device that would give the flotation machines a uniform feed of constant density.

The addition of acid may not be absolutely essential for all the ores of this district, although the local tests showed that a small quantity of acid is desirable, especially in cleaning the rougher concentrates. It was also found that when acid was not used sodium carbonate was beneficial in assisting the selective action of the oil or oil mixture, thus raising the grade of concentrate produced.

In conclusion, the problems to be solved in the application of flotation to the ores of this region may be summarized as follows: (1) The quantity of material available for treatment by flotation without further crushing; (2) the possibility of regrinding certain products from the various stages in concentration in the present milling practice; (3) the oils, oil mixtures, or frothing agents, best suited to the material to be treated; (4) the effect of the acid waters present in some of the mines of the district.

CHARACTER OF CONCENTRATES PRODUCED.

Zinc concentrates in the Joplin district assay 50 to 63 per cent metallic zinc, whereas the lead concentrates contain, on the average, 75 to 80 per cent lead. The average assay for the zinc concentrates for the district is about 57 to 58 per cent zinc, although the zinc concentrates produced from sheet-ground ore and from some of the new mines in Oklahoma and Kansas often run 2 to 3 per cent higher. The content of impurities varies. Some concentrates may contain 0.1 to 5 per cent iron and 0.1 to 2 per cent lead and some lots contain small proportions of lime. To show more fully the composition of the zinc concentrates the following analyses by W. G. Waring, of Webb City, Mo., are given in Tables 39 to 43:

TABLE 39.—*Results of analysis of composite sample of 3,800 shipment lots of Joplin zinc-blende concentrates sold in 1904.*

	Per cent.
Zinc.....	58.260
Cadmium.....	.304
Lead.....	.70
Iron.....	2.23
Copper.....	.049
Sulphur.....	30.42
Manganese.....	.01
Silica.....	3.95
Barium sulphate.....	.82
Calcium carbonate.....	1.88
Magnesium carbonate.....	.85
	99.773

TABLE 40.—*Analyses of products showing degree of separation attained in jigging zinc blende from calamine in mixed ore from the same mine. Wentworth district.*

Constituent.	Zinc blende.		Calamine.	
	Sample 1.	Sample 2.	Sample 1.	Sample 2.
	Per cent.	Per cent.	Per cent.	Per cent.
Total zinc (Zn).....	56.4	58.4	57.3	51.80
Zinc as ZnS.....	39.8	40.1	18.2	8.65
Zinc as ZnSiO ₃	16.6	18.3	33.1	43.15
Fe.....	2.2	3.4	1.5	1.3
CaO.....	0.45	0.15	0.42	0.40
MgO.....	Trace.	Trace.		
Pb.....	0.34	0.52		
Insoluble.....	11.67			
S.....				

TABLE 41.—*Analysis of product from mill treating calamine ore. Princeton district.*

	Per cent.
Zinc (Zn).....	40.0
Iron (Fe).....	0.8
Lime (CaO).....	4.05
Magnesia (MgO).....	1.44
Lead (Pb).....	
Insoluble.....	
Sulphur (S).....	1.43

TABLE 42.—*Analyses of lead concentrates from mill treating lead and zinc ore, Granby district.*

	Sample 1.	Sample 2.
Pb (total).....per cent..	74.3	80.0
Zn.....do...	5.2	5.16
Fe.....do...	4.2	3.9
PbCO ₃do...		6.5
PbSO ₄do...		
Insoluble.....do...	1.3	

A tabular summary showing the highest, lowest, and average assays of 426 shipments of lead and zinc concentrate from 34 chief producing mines in the Joplin district (including 7 Miami, Okla., mines) during January, February, and March, 1916, is presented in Table 43.

TABLE 43.—Data on 426 shipments of lead and zinc concentrates.

Mine No.	Zinc-blende concentrates (first-grade jig concentrates).						Lead (galena) concentrates.						Table concentrates (mixed ore and tailing ore).							
	High.			Low.			Average.			Lead content of jig concentrates.			Lead content of table concentrates.			Average content of—				
	Zn.	Fe.	Pb.	Zn.	Fe.	Pb.	Zn.	Fe.	Pb.	CaO.	High.	Low.	Average.	High.	Low.	Average.	Zn.	Fe.	Pb.	CaO.
1.....	63.4	2.4	0.4	62.8	1.8	1.50	63.09	2.04	0.25								53.40	5.00	0.97	
2.....	63.4	2.3	.85	62.0	1.7	1.20	62.53	1.96	.47								58.37	3.48	1.20	
3.....	61.1	1.9		60.2	1.3		60.65	1.55			79.2	75.7	78.18				56.43	2.51	1.30	
4.....	60.8	4.1	1.0	58.0	1.9	.50	57.78	3.29			81.8	80.3	81.00				57.04	2.45	2.50	
5.....	59.5	3.1	1.45	58.6	2.4	.18	59.03	2.64	.76								51.64	6.63	.85	
6.....	63.0	1.3	1.1	62.2	2.9	.40	62.62	1.05	.80								54.70	2.72	1.45	
7.....	63.4	2.1	1.68	62.2	1.1	.21	62.81	1.39	.45		77.2	75.3	76.67				57.70	2.21	.61	
8.....	63.5	2.4	.15	61.3	1.7	.04	62.46	2.02	.12								52.95	6.25	1.12	
9.....	61.6	1.7	.12	60.6	1.7		61.16	1.28	.06	1.55							53.50	2.40	.04	
10.....	55.0	5.4		52.6	3.5		53.90	4.30	.84								48.62	5.70	1.51	0.55
11.....	56.8	3.4		51.2	2.1		53.78	2.70	2.86	.47							49.30	3.40	3.83	
12.....	63.0	1.1		62.2	2.8	.50	62.72	2.90	.08				80.85				51.80	3.60	1.00	
13.....	57.9	3.9	1.1	56.0	2.9	1.30	57.14	3.29	1.97				81.80							
14.....	61.6	2.05		61.5	2.1		61.52	2.00	.25											
15.....	60.5	3.9	4.5	58.0	1.9	.30	58.96	2.72	.87											
16.....	58.9	2.2	1.9	58.0	1.4	1.00	57.68	1.55	1.41				82.80							
17.....	63.3	2.7	2.0	60.8	1.4	1.10	62.02	2.06	.16											
18.....	56.0	2.9	4.0	52.8	2.5	2.65	54.57	2.69	3.34											
19.....	62.2	1.6	4	60.8	1.3	1.0	61.82	1.48	.16											
20.....	61.3	3.3	.5	55.9	1.2	.15	59.05	1.97	.22											
21.....	56.0	3.8		54.0	2.9		54.93	3.31	1.67											
22.....	57.9	3.9		56.0	2.9		57.14	3.31	1.97		81.9	81.0	81.53							
23.....	54.9	7.8	.23	51.2	2.7	.03	53.41	6.97	.12	1.17										
24.....	60.3	2.6	.29	58.8	1.7	.03	59.63	1.80	.17	1.64										
25.....	55.7	2.4		53.0	1.6		54.32	1.94	1.15	1.15										
26.....	60.5	2.9		57.8	2.1		59.85	2.46												
27.....	47.6	13.8	1.0	36.7	7.2	.36	42.10	10.75	.62	.32										
28.....	57.7	2.3	1.0	55.8	2.1	.50	56.31	2.16	.65											
29.....	59.4	6.2	1.4	52.7	1.5	.80	56.98	2.80	1.12											
30.....	59.7	3.1		59.25	2.5		59.25	2.75	.37											
31.....	61.7			48.1			50.08													
32.....	61.7			38.05			38.05													
33.....	61.7			32.0			32.0													
34.....	61.7			42.80			42.80													

a Cadmium.

HANDLING OF CONCENTRATES AND POSSIBLE CAUSES OF DEPRECIATION IN WEIGHT DURING SHIPMENT.

The finished products from the mills are stored in bins just outside of the mill. After the concentrate has been sold it is shoveled into wagons and hauled to the shipping point, the hauling and freight charges being paid by the smelter or buyer.

It would be practically impossible to determine definitely the loss in weight of lead and zinc concentrates from the time they leave the mine or mill until they reach their destination, because of the many factors that may cause such loss.

The chief cause of this depreciation in weight is loss of moisture. The moisture loss, however, depends on the kind of ore from which the concentrates were derived, whether for zinc concentrates, from silicate, carbonate, or sulphide ore; whether the concentrates are coarse or fine; the moisture content at the start; the distance and time in transit; the weather conditions; and the season of the year in which the shipment is made. In winter the concentrates hold moisture longer than in summer. Shrinkage in weight through loss of moisture is generally more marked in fine than in coarse concentrates, partly because of the usually higher moisture content before shipment.

From a number of cars of medium coarse concentrates of zinc sulphide shipped during the months of December and January from a certain mine to the smelter a distance of about 440 miles, the depreciation in weight by evaporation of moisture in transportation was found to be 1.4 per cent. This difference no doubt would have been greater if the concentrates had been shipped during the summer. Moreover, the loss in moisture depends on the time the concentrate is in transit from the mill to the smelter and the weather conditions en route. The following averages of moisture content of zinc concentrates, figured from 10 samples for each month, during the fiscal year 1914 to 1915 as shipped to the smelter, may be of interest:

Average moisture content of zinc concentrates for 12 months.

	1914.	Moisture content, per cent.
July.....		3.07
August.....		3.39
September.....		3.16
October.....		3.56
November.....		3.52
December.....		3.53
	1915.	
January.....		3.91
February.....		4.22
March.....		3.30
April.....		3.69
May.....		3.93
June.....		3.47

NOTE.—A great deal of rain fell during the months of May and June, 1915, in contrast to the fairly dry month of June, 1914, when the average moisture content was 2.91.

Other causes of loss of weight in the shipment of concentrates from the mine or mill to the smelter are in hauling the material from the mill to the shipping cars, in loading and unloading the cars, and during rail transportation. Some ore buyers have stated that the depreciation in weight during transportation from causes other than the loss of moisture may amount to 0.5 per cent or less, whereas others claim that the loss is slight and that the net weights of the concentrates at the mine or mill and at the smelter usually check in the long run.

Also differences in weights have been found to be due to the use of inaccurate scales either at the mine or mill or at the smelter. To avoid difficulties from lack of agreement in the weight as determined from the scales at the mine or smelter and as determined by the railroads, the cars should always be weighed before loading, as the tare of a car will change in time through wear or through repairs, etc.

The preceding statements show how difficult it would be to obtain exact data regarding the depreciation in weight of concentrates during transit. The only method of procedure to follow in obtaining such data would be to take the matter up directly with the different smelters and from their figures calculate the differences in net weight of mineral content over a period of two or three years; and then to distribute this loss, as nearly as possible, to the various causes along the entire route of transportation.

Depreciation or change in assay value during transportation owing to chemical reaction of the ore or to any foreign material being mixed with the ore, is believed to be slight, if any.

DISPOSAL OF TAILINGS.

Usually throughout the region the tailings from the mills are piled near the mills by means of elevators and launders. As a pile of tailings increases in height the tailings from the launder are directed into the boot of another elevator and raised again as shown in Plate XV, A. Other means have been tried, such as belt conveyors and trams, but elevators seem to find favor among most of the operators.

Except for a few of the older and richer piles, the tailings are rarely retreated. This is especially true of tailings from the sheet-ground mines where the lead and zinc content of the ore is relatively low. Tailing piles of any considerable size are, as a rule, purchased by the railroads for use as road ballast. The tailings are usually loaded by means of steam shovels, as shown in Plate XVI, or the tailings from the mill may be permitted to flow directly through launders into railroad cars.



A. METHOD OF DISPOSING OF TAILINGS.



B. CONVEYING TAILINGS INTO MILL WITH BUCKET ELEVATOR FOR RE-TREATMENT.



LOADING TAILINGS FOR RAILROAD BALLAST.

RETREATMENT OF TAILINGS.

In the retreatment of tailings, certain conditions are essential to insure a good recovery of the mineral content. A few tailing piles have yielded good profits, but many piles have been retreated without sufficient testing previous to retreatment to determine the character of the material and the mineral recovery possible. Screen analyses of representative samples and assays of the screen products are necessary in determining the value of a tailing pile and the kind of equipment necessary for obtaining the best results. Screen analyses and assays of samples taken from many different tailing piles throughout the district have shown that the recoverable values were in the finer size material, whereas others, where the ore treated was chatty, showed the values in the coarser sizes. Screen analyses of tailings for retreatment will show whether the tailings need fine grinding or whether only a system of screening will be necessary.

Chemical analyses of various screen products of tailings taken from the interior of tailing piles have shown the presence of zinc in the form of sulphate. This is especially true where the mine waters have been acid, containing free sulphuric acid, and the tailing piles have been standing for several years. Although not enough tests were made to determine the proportion of the total zinc in the tailings as sulphate or as oxide, it would be well to determine by thorough sampling and assaying the form in which the zinc is present, especially for tailings from mines where the mine water has been acid and for tailing piles which show marked oxidation on the surface. Zinc sulphate is not only soluble in water but its specific gravity is less than that of flint. Some of the zinc may have been leached out of the tailing pile. Such preliminary precautions might save an operator the investment of a tailing mill.

Plates XV, *B*, and XVII show two methods of bringing the tailings into the mill for retreatment. The method shown in Plate XVII is the one most commonly used.

Screen analyses and assays of screen products of samples taken from fairly rich and relatively old tailing piles are shown in Tables 44 to 46 following. The tailing piles represented in Tables 44 and 45 were being retreated, but not more than 1.5 tons of concentrate, on the average, per 100 tons of tailings treated was being produced in either mill, which shows that poor zinc recoveries were being made. Finer grinding should have been done and the mills should have been properly equipped for treating the fine material. Table 46 represents a pile that would not pay to retreat under present conditions.

TABLE 44.—*Screen analyses of representative samples taken from tailing pile in West Joplin district.*

A. SAMPLES TAKEN FROM TOP OF PILE.

Size of mesh.	Size of screen openings.		Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc.
	Mm.	Inch.			
3	6.680	0.263	2.85	2.4	1.9
10	1.651	.065	64.9	3.4	62.1
35	.833	.0328	12.65	2.9	10.3
200	.074	.0029	13.95	4.6	18.2
<200	.074	.0029	5.65	4.7	7.5
Total or average.....			100.00	3.55	100.0

B. SAMPLES TAKEN FROM MIDDLE OF PILE.

3	6.680	0.263	6.0	1.7	2.5
10	1.651	.065	70.4	4.5	78.25
35	.833	.0328	10.55	2.7	7.06
200	.074	.0029	9.15	3.0	6.75
<200	.074	.0029	3.9	5.7	5.45
Total or average.....			100.0	4.05	100.00

C. SAMPLES TAKEN FROM BOTTOM OF PILE.

3	6.680	0.263	12.9	2.2	13.1
10	1.651	.065	56.25	2.3	59.4
35	.833	.0328	12.1	1.7	9.45
200	.074	.0029	14.45	1.8	11.95
<200	.074	.0029	4.3	3.1	6.1
Total or average.....			100.00	2.17	100.00

TABLE 45.—*Screen analysis of representative sample taken from tailing pile in Galena district.*

Size of mesh.	Size of screen openings.		Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc.
	Mm.	Inch.			
3	6.680	0.263	22.95	2.0	18.65
10	1.651	.065	48.7	2.2	43.6
35	.833	.0328	10.5	2.5	10.7
200	.074	.0029	14.65	3.6	21.45
<200	.074	.0029	3.2	4.3	5.6
Total or average.....			100.00	2.45	100.00

TABLE 46.—*Screen analysis of representative sample taken from tailing pile in Prosperity district.*

Size of mesh.	Size of screen openings.		Percentage of weight.	Zinc assay, per cent.	Percentage of total zinc.
	Mm.	Inch.			
3	6.680	0.263	24.7	0.9	24.9
10	1.651	.065	45.2	.9	45.5
35	.833	.0328	12.15	.4	5.45
200	.074	.0029	14.9	1.0	16.7
<200	.074	.0029	3.05	2.2	7.45
Total or average.....			100.00	0.89	100.00



DRAG-LINE METHOD OF BRINGING TAILINGS TO MILL FOR RE-TREATMENT.

LABOR CONDITIONS.

Nearly all the miners and surface men in the Joplin district are American born. As a rule they are efficient and intelligent, and readily pick up new ideas whenever they are given the opportunity.

The shift for underground work in the mines is 8 hours, whereas the mill shifts are 10 hours each. When a mine has been well developed a single underground shift can often break enough ore to keep the mill running during two 10-hour shifts. A few of the larger companies are now running their mills three 8-hour shifts rather than two 10-hour shifts. It is believed that this plan, if followed, will in many cases make for greater efficiency in both mining and milling.

In 1916 the average scale of wages paid to the miners in this district was based on a sliding-scale basis of a 25-cent increase or decrease for every \$10 increase or decrease in the average base price for 60 per cent zinc concentrates covering a period of one month. The following table shows the wages paid to the men, according to their occupation, based on the average price for 60 per cent zinc concentrates between \$80 and \$90 per ton.

TABLE 47.—*Sliding scale of wages based on an \$80 to \$90 base price for 60 per cent zinc concentrates.^a*

Underground men:	
Powder man.....	\$4. 00
Powder man helper.....	3. 50
Tub hooker.....	4. 00
Drill man.....	3. 75
Drill man helper.....	3. 25
Drill man, tripod.....	4. 00
Drill man helper, tripod.....	3. 50
Trackman.....	3. 75
Trackman helper.....	3. 25
Tub runner.....	2. 75
Bruno man ^b	2. 75
Roof man.....	3. 75
Shovelers, 9 to 10 cents per tub (1,000 pounds), average wage per day..	4. 00—4. 50
Pump man.....	3. 25
Surface men:	
Hoist man, shaft.....	4. 00
Hoist man, tram.....	2. 75
Cull hand.....	2. 80
Crusher feeder.....	3. 20
Jig-man helper.....	3. 75
Sludge man (running tables):	
10-hour shift.....	3. 20
8-hour shift.....	2. 95
Engineer.....	3. 75
Roustabout.....	2. 80

^a Based on figures given in the Engineering and Mining Journal, vol. 101, 1916, p. 751.

^b Term for miner who pulls down fine ore piled in a stope.

COST DATA.

To obtain average detail costs of mining and milling operations in this district would be difficult, as each company has its own system of keeping costs, and the mining conditions differ somewhat. Most of the larger companies keep detailed cost accounts, whereas others, usually at the smaller mines, keep only general costs. Three examples of itemized cost data are given in Tables 48 to 50. These cost data are for relatively large mines that have been operating for some time and can, therefore, be considered somewhat lower than the average throughout the district. In the new zinc fields that have recently been opened up near Tar River and Picher, Okla., the mining and milling costs are considerably higher than those in the tables, ranging from \$1.50 to \$2.50 per ton; but due to the higher lead and zinc content the costs per ton of concentrates produced are much lower, at present averaging \$20 to \$30 per ton.

TABLE 48.—*Mining and milling costs of a representative sheet-ground mine in the Webb City-Carterville district.*

Item.	1914	1915	1916
Mining costs [costs per ton of ore mined]:			
Ground boss.....	\$0.0102	\$0.0117	\$0.0119
Drilling.....	.1927	.2608	.2735
Blasting.....	.1721	.1867	.1782
Roof trimming.....	.0111	.0160	.0182
Loading.....	.1509	.2125	.2317
Conveying to shaft.....	.0705	.1106	.1427
Hoisting.....	.0367	.0405	.0487
Lighting.....	.0070	.0082	.0082
Miscellaneous (superintendence, insurance, etc.).....	.0209	.0403	.0711
Total mining cost.....	.6721	.8843	.9822
Milling costs [costs per ton of ore milled]:			
Culling.....	.0195	.0120	.0107
Crushing.....	.0448	.0322	.0376
Elevating.....	.0402	.0430	.0392
Jigging.....	.0318	.0358	.0387
Tabling.....	.0364	.0420	.0372
Water.....	.0070	.0069	.0063
Power.....	.0279	.0291	.0306
Miscellaneous (superintendence, insurance, etc.).....	.0202	.0226	.0257
Total milling cost.....	.2278	.2226	.2260
Total mining and milling costs [costs per ton of ore mined]:			
Mining.....	.6721	.8843	.9822
Milling.....	.2249	.2214	.2255
Pumping.....	.0587	.0507	.0443
Miscellaneous.....	.0623	.0307	.0240
General administrative.....	.0398	.0252	.0435
Total mining and milling.....	1.0478	1.2123	1.3195
Total cost per ton of concentrates produced.....	33.3750	44.9813	51.9855
Quantity of concentrates produced, per 100 tons of ore mined and milled:			
Lead concentrates.....	Tons. 1.0109	Tons. 0.9423	Tons. 0.9686
Zinc concentrates.....	2.1285	1.7528	1.5886
Total concentrates.....	3.1394	2.6951	2.5572

TABLE 49.—*Mining and milling costs of a sheet-ground mine in which mining consists principally in "taking up stope."*

[Cost per ton of ore mined.]

Item.	1914	1915	1916
Mining costs:			
Labor, without shoveling.....	\$0.2621	\$0.1657	\$0.2558
Labor, shoveling.....	.1362	.1713	.1802
Salaries.....	.0215	.0117	.0132
Explosives.....	.2036	.0620	.1464
Fuel.....	.0537	.0348	.0471
Liability insurance.....	.0159	.0146	.0196
Compensation for injuries to employees.....	.0028	.0111	.0020
Water.....	.0077	.0043	.0047
Lubricating oil.....	.0073	.0041	.0046
Repairs, supplies, and sundries.....	.0410	.0315	.0463
Total mining cost.....	a. 7518	.5411	.7204
Milling costs:			
Labor.....	.0825	.1062	.1293
Salaries.....	.0215	.0117	.0132
Hard iron.....	.0327	.0266	.0297
Belting.....	.0228	.0063	.0112
Fuel.....	.0253	.0190	.0203
Liability insurance.....	.0033	.0046	.0053
Compensation for injuries to employees.....	.0001	.0008	.0005
Water.....	.0077	.0044	.0047
Lubricating oil.....	.0028	.0022	.0017
Repairs, supplies, and sundries.....	.0309	.0295	.0436
Total milling costs.....	.2301	.2133	.2608
Total mining and milling costs.....	a. 9819	.7544	.9812
Total cost per ton of concentrates produced.....	28.360	50.1940	55.3083
Quantity of concentrates produced per 100 tons of ore mined and milled:			
Lead concentrates.....	<i>Tons.</i> 0.5922	<i>Tons.</i> 0.0785	<i>Tons.</i> 0.1014
Zinc concentrates.....	3.1326	1.4244	1.6728
Total concentrates.....	3.7248	1.5029	1.7740

* In 1914 both heading and stope were being mined with a much smaller tonnage, hence costs were relatively higher than in 1915.

TABLE 50.—*Mining and milling costs of a representative mine in the Commerce district, Okla.*

[Costs per ton of ore mined.]

Item.	1914	1915	1916
Ground labor.....	\$0.2885	\$0.4001	\$0.4667
Power.....	.0725	.0929	.1368
Timber.....	.0131	.0028	
Powder.....	.1267	.1591	.1513
Mill labor.....	.1328	.1896	.2451
Mill fuel.....	.0218	.0222	.0362
Mill supplies.....	.1117	.1078	.1178
Shop labor.....	.0427	.0478	.0568
Shop supplies.....	.0594	.0633	.0645
Top labor.....	.0758	.0841	.1161
Pumping.....	.0140	.0202	.0291
General expenses (including overhead).....	.0475	.1265	.1599
Total mining and milling cost.....	1.0065	1.3164	1.5803
Total cost per ton of concentrates produced.....	18.8052	30.4910	39.6203
Quantity of concentrates produced per 100 tons of ore mined and milled:			
Lead concentrates.....	<i>Tons.</i> 1.6450	<i>Tons.</i> 1.6582	<i>Tons.</i> 1.2078
Zinc concentrates.....	3.7066	2.6588	2.7757
Total concentrates.....	5.3516	4.3170	3.9836

MARKETING LEAD AND ZINC ORES.

The schedules for the purchase of lead and zinc ore in the Joplin district and the methods of marketing the ores differed somewhat from the schedules and methods employed in the Western States. The producer is paid by the ton of concentrate, the ore being purchased outright at the mine. The hauling and freight charges are paid by the smelter. The price paid for the various grades of concentrates varies according to a sliding scale based on the price paid for concentrates of a standard grade, which is known as the base price. As the concentrates are sold on the open market, there is more or less competition among the smelter representatives in the district who bid on the various lots offered for sale by the producers.

The schedules of basis on which the different kinds of ore are purchased are (1) 60 per cent metallic zinc for sphalerite or zinc-blende concentrates, (2) 40 per cent metallic zinc for concentrates of the oxidized zinc ores, the carbonate and silicate, and (3) 80 per cent metallic lead for the galena concentrates. The base price depends largely upon the state of the spelter market and is determined during the latter part of each week.

As the different grades of concentrates produced from the many mines throughout the district vary, the prices received per ton of concentrates, dry weight, are determined by the grade of concentrates and the content of impurities. Thus for zinc-blende concentrates, if the assay varies from the basis, \$1 is added to or deducted from the base price for each 1 per cent of zinc above or below 60 per cent. Penalties are imposed for certain impurities if present. Usually for each per cent of iron above 1 per cent, \$1 is deducted from the base price. Where the percentage of iron is high this penalty is often greatly reduced, if imposed at all. Lead and lime are penalized, usually by deducting a certain amount from the base price per ton of concentrates. This results in two or three different grades of zinc concentrates, each with its base price. For example, a zinc-blende concentrate containing 0.5 per cent or more lead would no longer be classed under the standard grade, but under a grade with a lower base price.

Because of the great demand for the better grades of spelter since the war began, the smelters have been somewhat more exacting as to impurities in zinc concentrates and have lowered the allowable percentages of impurities for the different grades. Other undesirable impurities in zinc concentrates are barite, fluorspar, and bitumen, but these are not frequently found with the ores of the Joplin district. Sorted ore, such as lump galena ore and the oxide ores of zinc, is usually sold on a flat bid, the ore buyer inspecting the ore and making an offer of so much per ton for it.

HEALTH CONDITIONS IN THE MINES.

In a metal mine the conditions affecting the health of the miner depend largely on the material mined, the depth of the mine, the gases present, and the ventilation. In the Joplin district the mines are shallow, injurious gases such as are found in coal mines or deep metal mines are lacking, and the ventilation is generally good.

VENTILATION.

The adequate ventilation is largely a result of thorough inspection by the State mine inspectors, who zealously enforce the law compelling operators to sink shafts, drill air holes, or cut drifts to openings, wherever needed to protect the health of the miners. Fans are used where natural ventilation is insufficient or can not be utilized.

WASH AND CHANGE HOUSES.

In many mines the ground underfoot is wet, so that the men, after the shift, leave the mine workings with clothing more or less wet. Most of the mines have change houses with individual lockers, a place for hanging and drying wet clothes, a large wash sink with running hot and cold water, and a few with shower baths. As many of the men drive or walk to and from the mines the change houses are necessary and are largely used. Although in general these buildings are small and not elaborate, they serve the purpose of allowing the clothes to dry after each shift, and guard the men against unnecessary exposure. At mines where there are no change houses the men are subjected, especially during the winter months, to a sudden change of temperature at the surface, with the consequent risk of severe colds. Although the men become hardened to these conditions, the constant exposure day after day gradually reduces their vitality, and precautions to prevent such conditions are highly desirable.

ABATEMENT OF ROCK DUST.

In the mines, especially the sheet-ground mines, that are practically free from water and dampness, more or less fine dust from the drilling and blasting may be carried in the air. This dust, if inhaled by the miner every day, may in time cause lung diseases. To quote from the preliminary report^a on this district:

The extent of harm done by inhaling the dust in metal mines is not definitely known, but it is believed that much of the lung trouble often known as "miners' consumption" has been caused by it. Some of the operators in the district lessen the injurious effect of rock dust by dampening the faces, but this is done at few mines.

^a Wright, C. A., Mining and treatment of lead and zinc ores in the Joplin district, Missouri, a preliminary report: Tech. Paper 41, Bureau of Mines, 1913, p. 40

This wetting of the face, however, should be done at all mines where rock dust is prevalent, for it would without doubt prevent much sickness and spreading of disease. In this connection it may be stated that the Bureau of Mines, in cooperation with the Bureau of Public Health, is to investigate a number of problems relating to the hygiene of mines and to the mining industry, the investigations being conducted with regard to scientific as well as administrative problems and including a study of the prevalence and prevention of lung diseases among miners.

Following a preliminary report ^a of these investigations, published in 1915, a more detailed study was made of the harmful effect of the finely divided, siliceous rock dust on the miner's lungs when it is suspended in the air in the working drifts, and also with regard to other sanitary conditions at the mines as well as in the homes of the miners. Dr. Lanza, of the Public Health Service, spent several months making physical examinations of the miners and their families to determine the prevalence of pulmonary disease. The complete report of the investigations has been published in Bulletin 132^b of the Bureau of Mines, to which the reader is referred for further details.

As a result of the investigations and the enforcement of the new mining laws and the careful supervision of the State mine inspectors, there has been a marked gain in abating the rock dust in the mines and in improving sanitary conditions. Also the miners themselves have shown willingness to cooperate with the mining companies.

RÉSUMÉ.

In view of the fact that the main object of every mining operation is to obtain the greatest profits from the expenditures made, certain conditions must constantly be considered. The operator should not only attempt to obtain the most efficient working conditions with the equipment and labor on hand, but should also investigate the possibilities of improvements that will result in a greater saving and in increased profits, even if they involve changes in the methods of mining and milling at no small cost. Although there has been a marked improvement in the saving of mineral the past few years by improved mill construction and more efficient operation and equipment, at many properties a greater mineral recovery and increased profits could be realized. As the conditions at any two mines are not exactly alike, each mine or mill must be considered by itself. Likewise any improvements that could be made for increasing the working efficiency in labor, time, and production, must be determined under the existing conditions at each mine separately. Hence a few of the main points that might prove worthy of more thorough

^a Lanza, A. J., and Higgins, Edwin, Pulmonary disease among miners in the Joplin district, Missouri, and its relation to rock dust in the mines: Tech. Paper 105, Bureau of Mines, 1915, 48 pp.

^b Higgins, Edwin, Lanza, A. J., Laney, F. B., and Rice, G. S., Siliceous dust in relation to pulmonary disease among miners in the Joplin district, Missouri: Bull. 132, Bureau of Mines, 1917, 112 pp.

investigation by the operators at many of the mines and mills, as brought out more or less in detail in the preceding pages of this report, are given here.

The proper grade and quantity of explosive, and also the proper strength of cap, to be used in breaking ground, should be determined according to the character of the ore-bearing ground and the product to be obtained. The increased effectiveness of powder by the use of stemming should be determined as to the quantity of ore broken. A saving in the use of powder would be an important item in reducing the mining costs.

To follow a system of mining that leaves the working face more accessible and in the best condition for subsequent drilling and blasting is important. The drill man or mine foreman should consider and try to foresee as far as practicable the form in which the face will be left after each round of shots.

The best method of hauling the ore from the working face to the shaft for saving time, labor, and costs should be ascertained.

The only sorting in the mines of this district worthy of mention is the culling of large boulders of waste rock. Because of the large quantities of waste mined with the ore, especially in the sheet-ground mines, where in many places the ore is in "runs" with a comparatively thick layer of barren rock between, it is reasonable to assume that some system of hand sorting, underground or in the mill, could be used successfully. The discarding of a larger proportion of waste rock would result in a relatively higher metal recovery in the mills, save power, reduce wear and tear on crushing machinery, elevator belts, and other equipment, and increase the relative capacity of the mill.

At the mills sampling and mill testing should be done to find out what results are being obtained. To be of value, however, the sampling should be careful and systematic. The sampling of tailings without including the overflow water from settling tanks in determining the total metal loss of a mill, an oversight frequently observed, is of little value. Screen analyses of the samples to determine what sizes contain the valuable minerals are important, as a more definite idea can be obtained as to how and where further savings may be effected.

In jigging, it would seem advisable to deslime all material before it passes to the rougher jigs. Where the ore is chatty and a large quantity of middlings is produced, the middlings should be reground and treated separately over "chat" jigs rather than crushed and returned to the first rougher jigs, as practiced in some mills.

Desliming of the feed and separate treatment of the middlings can be applied to the treatment of sands on tables equally as well as in

jigging. Roughing and cleaning tables could also be used to advantage in some of the larger mills of the district.

The possibilities of flotation are well worth investigation by every operator, as a small flotation unit might increase the mineral recovery 10 to 20 per cent.

In addition to efficiency in the mining and milling of the ores, health and safety conditions should be given more attention. Each operating company should see that in its mine or mill due effort is made to effect any improvement that will better the health and safety conditions. Such subjects as the safe use of explosives, mine timbering, fire protection, safety appliances, and sanitation in and about the mines, require careful study and should be just as vital to the mine operator as the number of tons of ore mined and treated each day.

At many of the mines and mills excellent results are being obtained. It is hoped, however, that every operator will endeavor to effect greater saving, so far as is profitable, and take advantage of any experiments made by others that would improve his own practice if applied.

MISCELLANEOUS DATA.

The number of zinc-smelting retorts in the United States at the end of 1916, with comparative figures for 1915, are given in Table 49. Monthly prices for lead and spelter for these years are shown in Tables 50 and 51.

TABLE 51.—Zinc-smelting capacity of plants in the United States.^a

(Number of retorts at end of 1915 and 1916.)

Name.	Situation.	1915.	1916.
American Spelter Co.....	Pittsburg, Kans.....	896	^b 896
American Steel & Wire Co.....	Donora, Pa.....	3, 648	9, 120
American Zinc and Chem. Co.....	Langeloth, Pa.....	3, 648	7, 296
American Zinc Co. of Illinois.....	Hillsboro, Ill.....	4, 000	4, 864
American Zinc, Lead and Smg. Co.....	Dearing, Kans.....	4, 480	4, 480
American Zinc, Lead and Smg. Co.....	Caney, Kans.....	6, 080	6, 080
Arkansas Zinc and Smelting Corp.....	Van Buren, Ark.....		2, 400
Athletic Min. and Smelting Co.....	Fort Smith, Ark.....		(c)
Bartlesville Zinc Co.....	Bartlesville, Okla.....	5, 184	7, 488
Bartlesville Zinc Co.....	Blackwell, Okla.....		8, 800
Bartlesville Zinc Co.....	Collinsville, Okla.....	10, 080	13, 440
Chanute Spelter Co. ^d	Chanute, Kans.....	1, 280	1, 280
Cherokee Smelting Co.....	Cherokee, Kans.....	896	^e 896
Clarksburg Zinc Co.....	Clarksburg, W. Va.....	3, 648	3, 648
Collinsville Zinc Co.....	Collinsville, Ill.....	^b 1, 536	1, 984

^a From Engineering and Mining Journal, vol. 103, Jan. 6, 1917, p. 25.

^b No report received; entered the same as previous year.

^c Under construction.

^d Operated by American Metal Co. in 1915.

^e Idle.

TABLE 51.—Zinc-smelting capacity of plants in the United States—Continued.

Name.	Situation.	1915.	1916.
Eagle-Picher Lead Co.	Henryetta, Okla.		3,000
Edgar Zinc Co.	Carondelet, Mo.	2,000	2,000
Edgar Zinc Co.	Cherryvale, Kans.	4,800	4,800
Fort Smith Spelter Co.	Fort Smith, Ark.		2,560
Granby Mining and Smltg. Co. ^a	Neodesha, Kans.	3,760	3,760
Granby Mining and Smltg. Co. ^a	East St. Louis, Ill.	3,220	4,864
Grasselli Chemical Co.	Clarksburg, W. Va.	5,760	5,760
Grasselli Chemical Co.	Meadowbrook, W. Va.	8,592	8,544
Grasselli Chemical Co.	Terre Haute, Ind.		(b)
Hegeler Zinc Co.	Danville, Ill.	3,600	5,400
Henryetta Spelter Co.	Henryetta, Okla.	600	3,000
Illinois Zinc Co.	Peru, Ill.	4,640	4,640
Iola Zinc Co.	Concreto, Kans.	660	c 660
Joplin Ore and Spelter Co.	Pittsburg, Kans.	1,440	d 1,792
J. B. Kirk Gas and Acid Co. ^e	Iola, Kans.	3,440	3,440
Kusa Spelter Co.	Kusa, Okla.	3,710	3,720
La Harpe Spelter Co.	Kusa, Okla.	(f)	4,000
Lanyon Smelting Co.	Pittsburg, Kans.	448	448
Robert Lanyon Zinc and Acid Co.	Hillsboro, Ill.	1,840	3,200
Lanyon-Starr Smelting Co.	Bartlesville, Okla.	3,456	3,456
Matthiessen and Hegeler Zinc Co.	La Salle, Ill.	6,168	6,168
Mineral Point Zinc Co.	Depue, Ill.	9,068	9,068
Missouri Zinc Smelting Co.	Rich Hill, Mo.		4,448
National Zinc Co.	Bartlesville, Okla.	4,260	4,970
National Zinc Co.	Springfield, Ill.	3,200	3,800
Nevada Smelting Co.	Nevada, Mo.	672	672
New Jersey Zinc Co. of Penn.	Palmerton, Pa.	6,720	7,200
Oklahoma Spelter Co. ^g	Kusa, Okla.	(f)	1,600
Owen Spelter Co. ^h	Caney, Kans.	1,280	1,920
Pittsburg Zinc Co. ^h	Pittsburg, Kans.	910	910
Prime Western Spelter Co.	Gas City, Kans.	4,868	4,868
Quinton Spelter Co.	Quinton, Okla.		1,340
Sandoval Zinc Co.	Sandoval, Ill.	896	672
Tulsa Fuel and Manufacturing Co.	Collinsville, Okla.	6,232	6,232
United States Smelting Co.	Altoona, Kans.	3,960	4,600
United States Smelting Co.	Checotah, Okla.		4,480
United States Smelting Co.	La Harpe, Kans.	1,924	1,928
United States Zinc Co. ⁱ	Sand Springs, Okla.	5,680	8,000
United States Zinc Co.	Pueblo, Colo.	2,208	1,984
United Zinc Smelting Corp.	Moundsville, W. Va.		(b)
Weir Smelting Co.	Weir, Kans.	(f)	448
Total		155,388	211,12

^a Subsidiary of American Zinc, Lead & Smelting Co., 1916.^b Under construction.^c No report received; entered the same as previous year.^d Idle latter part of 1916.^e Taken over by United States Smelting Co. in July, 1916.^f Was under construction in 1915.^g Operated by American Zinc, Lead & Smelting Co.^h Operated by American Metal Co. in 1915.ⁱ Formerly Tulsa Spelter Co.

TABLE 52.—*Monthly prices of lead at New York City, N. Y., for the years 1905–1916.*^a

[In cents per pound.]

	1905	1906	1907	1908	1909	1910	1911	1912	1913	1914	1915	1916
January.....	4.552	5.600	6.000	3.691	4.175	4.700	4.483	4.435	4.521	4.111	3.739	5.921
February.....	4.450	5.464	6.000	3.725	4.018	4.613	4.440	4.026	4.325	4.048	3.827	6.246
March.....	4.470	5.350	6.000	3.838	3.986	4.459	4.394	4.073	4.327	3.970	4.053	7.036
April.....	4.500	5.404	6.000	3.993	4.168	4.378	4.412	4.200	4.381	3.810	4.221	7.630
May.....	4.500	5.685	6.000	4.253	4.287	4.315	4.373	4.194	4.342	3.900	4.274	7.463
June.....	4.500	5.750	5.790	4.466	4.350	4.343	4.435	4.392	4.325	3.900	5.922	6.936
July.....	4.524	5.750	5.288	4.447	4.321	4.404	4.499	4.720	4.353	3.891	5.659	6.352
August.....	4.665	5.750	5.250	4.580	4.363	4.400	4.500	4.569	4.624	3.875	4.656	6.244
September.....	4.850	5.750	4.813	4.515	4.342	4.400	4.485	5.048	4.698	3.828	4.610	6.810
October.....	4.850	5.750	4.750	4.351	4.341	4.400	4.265	5.071	4.402	3.528	4.600	7.000
November.....	5.200	5.750	4.376	4.330	4.370	4.442	4.298	4.615	4.293	3.663	5.155	7.042
December.....	5.422	5.900	3.658	4.213	4.560	4.500	4.450	4.308	4.047	3.800	5.355	7.513
Average for year.....	4.707	5.347	5.325	4.200	4.273	4.446	4.420	4.471	4.370	3.962	4.628	6.856

^a From Engineering and Mining Journal, vol. 83, Jan. 5, 1907, p. 10; vol. 85, Jan. 4, 1908, p. 11; vol. 89, Jan. 8, 1910, p. 67; vol. 93, Jan. 6, 1912, p. 18; vol. 96, Jan. 6, 1915, p. 58; vol. 108, Jan. 6, 1917, p. 18.

TABLE 53.—*Monthly prices of spelter at St. Louis, Mo., for the years 1905–1916.*^a

[In cents per pound.]

	1905	1906	1907	1908	1909	1910	1911	1912	1913	1914	1915	1916
January.....	6.032	6.337	6.582	4.368	4.991	5.961	5.302	6.292	6.854	5.112	6.211	16.745
February.....	5.999	5.324	6.064	4.038	4.739	5.419	5.268	6.349	6.089	5.226	8.253	18.269
March.....	5.917	6.056	6.087	4.527	4.607	5.487	5.413	6.476	5.926	5.100	8.369	16.676
April.....	5.967	5.931	6.535	4.496	4.815	5.269	5.249	6.483	6.491	4.963	9.857	16.325
May.....	5.284	5.846	6.291	4.458	4.974	5.041	5.198	6.539	5.250	4.924	14.610	14.108
June.....	5.040	5.948	6.269	4.388	5.252	4.978	5.370	6.727	4.974	4.850	21.038	11.623
July.....	5.247	5.856	5.922	4.388	5.252	5.002	5.545	6.966	5.128	4.770	18.856	8.753
August.....	5.556	5.878	5.551	4.556	5.579	5.129	5.908	6.878	5.508	5.418	12.011	8.589
September.....	5.737	6.056	5.089	4.619	5.646	5.364	5.719	7.813	6.444	5.280	13.270	8.539
October.....	5.934	6.070	5.280	4.651	6.043	5.478	5.951	7.276	5.188	4.750	12.596	9.659
November.....	5.964	6.225	4.775	4.909	6.221	5.826	6.223	7.221	5.088	4.963	15.792	11.422
December.....	6.374	6.443	4.104	4.987	6.099	5.474	6.151	7.061	5.004	5.430	15.221	10.496
Average for year.....	5.730	6.048	5.812	4.578	5.352	5.370	5.606	6.799	5.504	5.061	12.064	12.634

^a From Engineering and Mining Journal, vol. 83, Jan. 5, 1907, p. 13; vol. 85, Jan. 4, 1908, p. 15; vol. 89, Jan. 8, 1910, p. 71; vol. 93, Jan. 6, 1912, p. 23; vol. 96, Jan. 6, 1915, p. 64; vol. 108, Jan. 6, 1917, p. 24.

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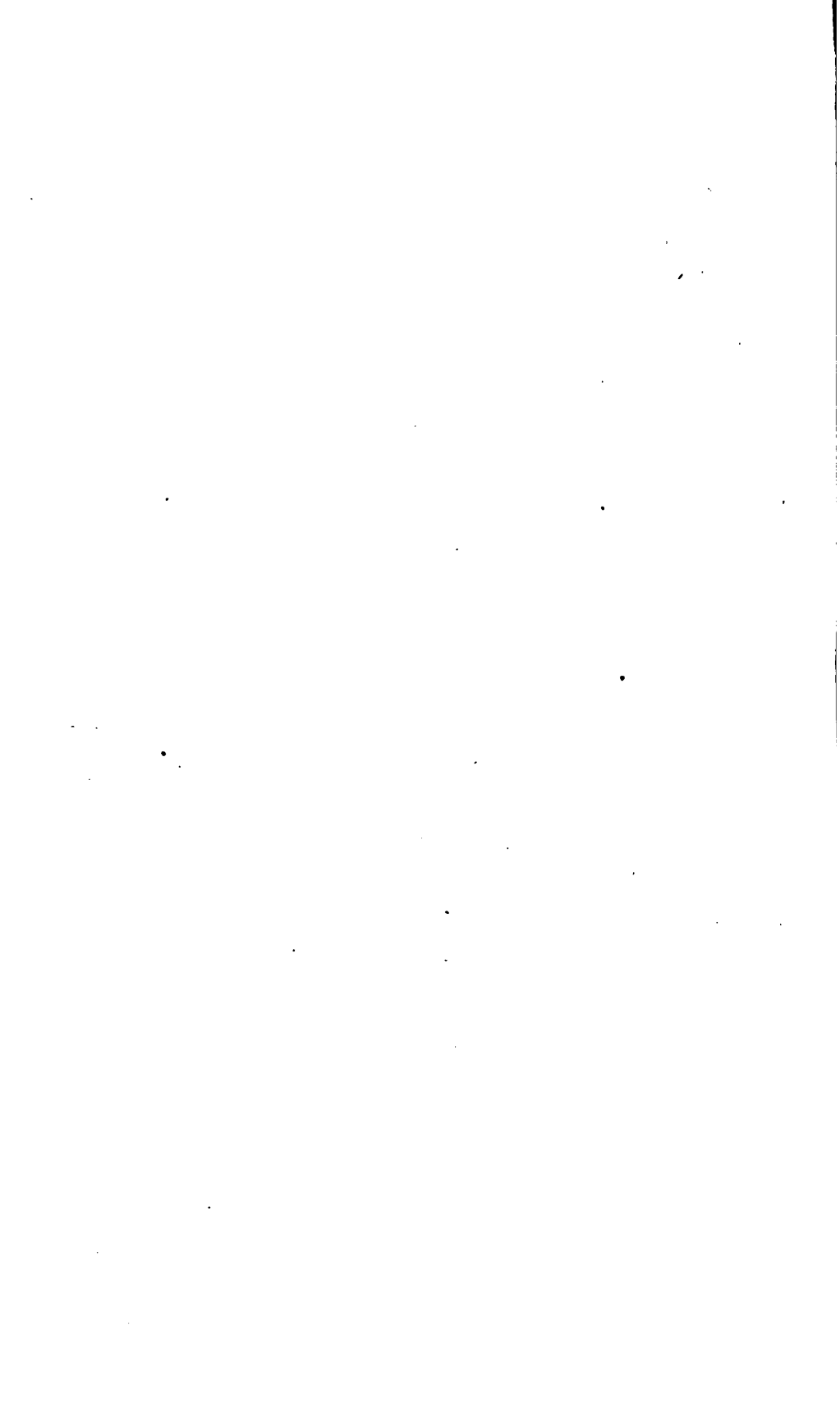
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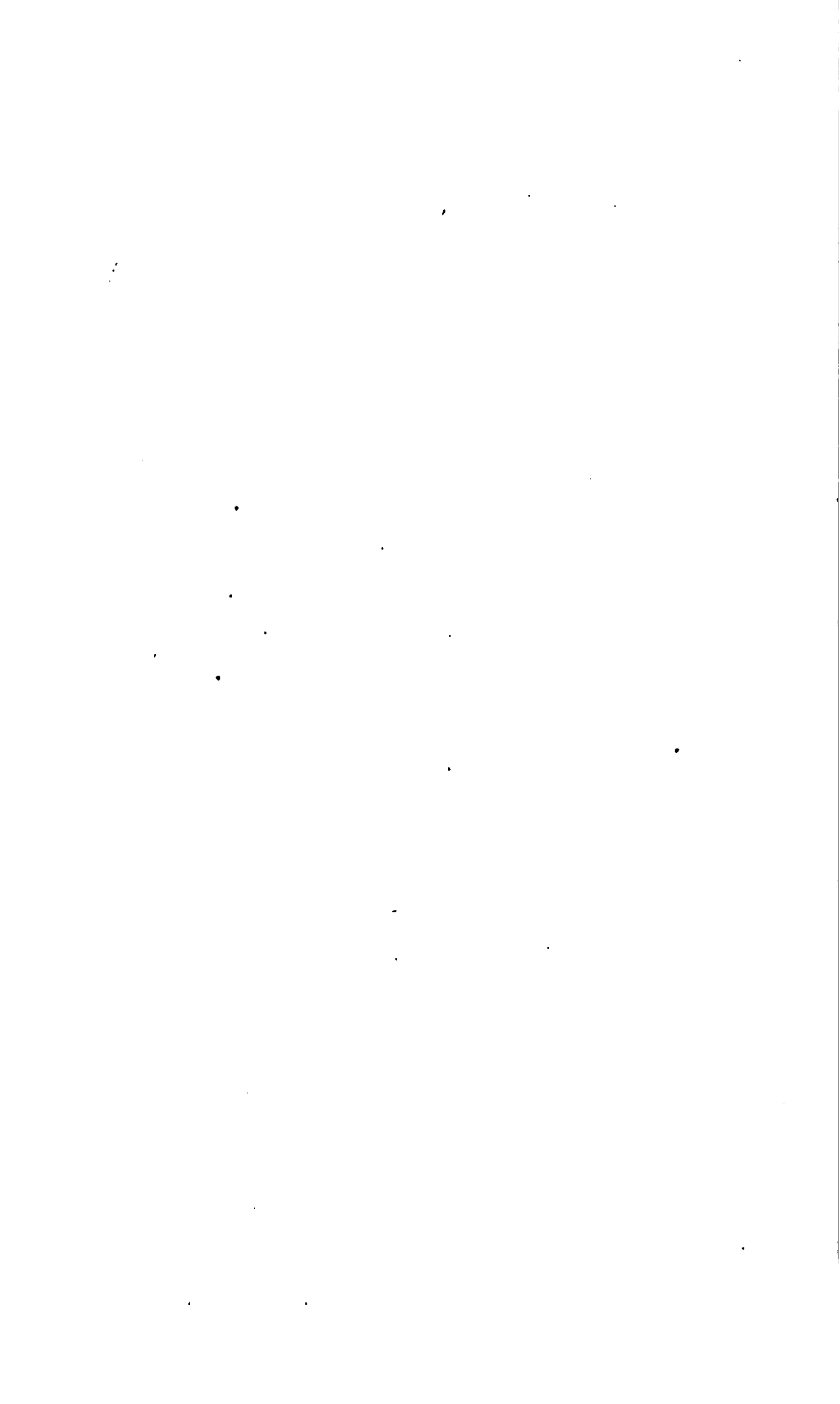
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DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

OIL-STORAGE TANKS AND RESERVOIRS

**WITH A BRIEF DISCUSSION OF LOSSES OF OIL
IN STORAGE AND METHODS OF PREVENTION**

BY

C. P. BOWIE



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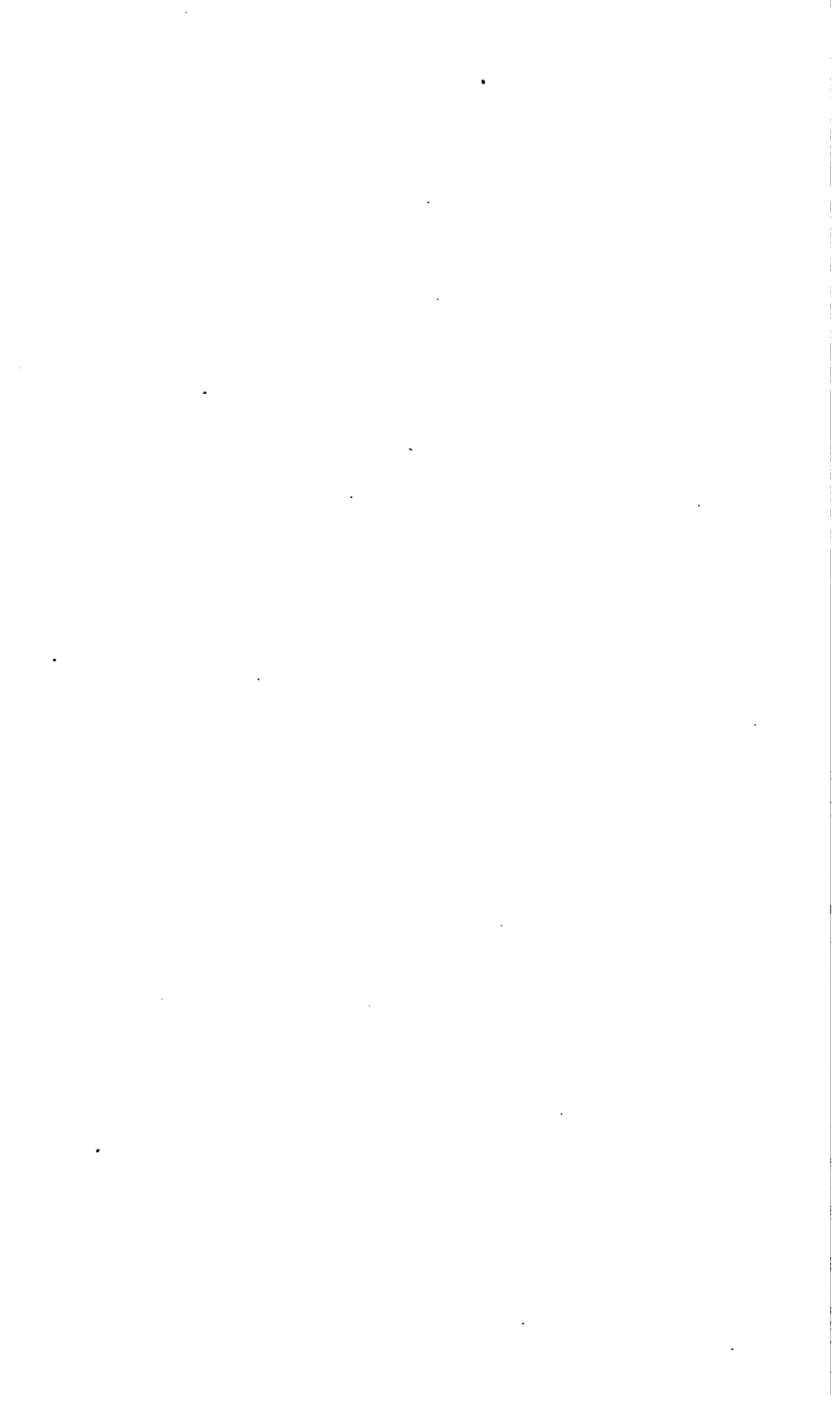
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OIL-STORAGE TANKS AND RESERVOIRS, WITH A BRIEF DISCUSSION OF LOSSES OF OIL IN STORAGE AND METHODS OF PREVENTION.

By C. P. BOWIE.

INTRODUCTION.

The Bureau of Mines has been conducting investigations with the view of determining the types of containers best adapted to the storage of oil. These investigations have shown that tanks composed wholly of steel give the best results. Where larger containers than it is feasible to build with steel are desired, concrete-lined reservoirs can be recommended for some grades of oil. Practically all such containers in use at present have wooden roofs and this type of construction is here described, although it is the belief of the writer that concrete roofs would be far more satisfactory in every way, and that the difference in cost between a concrete and a wooden roof would, as a rule, in a few years' time, be offset by a saving of oil and in cost of repairs and renewals.

TANK FARMS.

Storage facilities, whether steel tanks or reservoirs, are grouped together in what are known as "storage farms," some of which in the United States already have capacities in excess of 24,000,000 barrels. The ideal site for a farm is on ground that is comparatively level. Tanks, which are usually of 37,000 to 55,000 barrel capacity, are placed about 500 feet center to center, making the distance shell to shell about 400 feet. Each tank is then surrounded by a levee of sufficient height to hold the entire content of the tank, as shown on Plate I, A. In many instances these levees are circular and are themselves inclosed by a system of rectangular levees, built equidistantly between tanks, and dividing the whole farm into a system of checkerboard squares, the tanks being at the centers of the squares. Should any one tank catch fire it is thus isolated from its neighbors, and even though it may burn until its entire contents are consumed, if the levees are properly constructed it will do so without undue menace to the other tanks. As regards large reservoirs of 500,000 to 1,000,000 barrel

capacity, it is of course not practical to make the empounding area of a surrounding levee of sufficient size to hold the entire contents of the reservoir. Nevertheless, levees between reservoirs are advisable, and it is the practice to provide them. Levees are to be discussed more fully in a subsequent publication on fire protection to be issued by the Bureau of Mines.

SIZES OF STEEL TANKS.

Steel tanks for the storage of oil are manufactured in various sizes up to 55,000-barrel capacity. These have a diameter of 114 feet 6 inches and are 30 feet high. The usual size is the 55,000-barrel tank, although the Standard Oil Co., especially, is using great numbers of 37,000-barrel tanks. Tanks may be erected by contract or by the owners. However, for various reasons, the contract method is generally preferable. Under this method the company furnishes the grades on which the tanks are placed, and transportation of men and materials from the nearest railroad station; the contractors furnish the material, the railroad transportation, and all labor for erection.

TANK GRADES.

As to tank grades, it is well to note in passing that too much care can not be exercised in their preparation. They should be either all in fill or all in excavation and carefully graded to a true level surface. Plate I, *B*, shows the method of setting grade stakes. A tank built partly on filled ground and partly on excavated ground is in constant danger of being disrupted at the point where the foundation goes from cut to fill, as the filled part of the foundation will settle, whereas the part in cut ground probably will not, or at least not to so great a degree. Where it is necessary to build up a grade, the best policy is to fill the entire area to approximately the same depth, so as to insure as uniform settling as possible. It is good practice, also, unless the length of haul is excessive, to use the Fresno type of grader and to distribute the dirt in thin layers, allowing the stock to pass back and forth over the loose material as often as feasible. Puddling the grade in filled ground is also good practice.

As the bottom of the tank is composed of relatively thin plates, every precaution should be taken to prevent its deterioration from the corrosive action of alkaline salts contained in the soil upon which it rests, and also from similar chemical action by reason of water leaking through an imperfect seam and forming a puddle beneath the plates.

To eliminate as far as possible the detrimental effect of soil corrosion, several inches of oil sand is usually placed on the surface of the completed grade, or if this is impracticable, the completed surface is oiled, and the oil worked in 3 to 4 inches by rakes. The tank bot-



A. PART OF A TANK FARM UNDER CONSTRUCTION.



B. METHOD OF SETTING GRADE STAKES FOR FINISHING TANK GRADE TO TRUE LEVEL.



C. TANK BOTTOM IN COURSE OF CONSTRUCTION.

tom is thoroughly coated with a good quality of asphaltic or other rust-resisting paint before being lowered onto the grade. Plate I, C, shows a tank bottom in course of construction.

TANK SPECIFICATIONS.

Perhaps no better description of a steel tank can be given than that furnished by a recital of the specifications covering its erection and the material of which it is composed. Following are such specifications for a tank of 55,000-barrel capacity, of steel construction throughout, and of a type known to the trade as "gas-tight." This capacity of tank is considered to be the largest practicable for fabrication and construction in the field, and the type of steel tank best adapted to the storage of petroleum products so far as regards fire hazards and losses by seepage and evaporation.

SPECIFICATIONS FOR 55,000-BARREL ^a STEEL TANK WITH STEEL ROOF.

The _____, in itself or by its duly authorized representative, shall be referred to in these specifications as the "Company."

The firm undertaking to furnish and erect the tank herein described, in itself or by its duly authorized representative, shall be referred to in these specifications as the "Contractor."

At all times during the progress of the fabrication of the tank material or erection of the tank, the Contractor shall designate some person as his representative, to whom instructions may be given by a duly authorized representative of the Company.

(1) *Location*.—Tank to be erected upon foundation prepared by the Company.

(2) *Freight and hauling*.—The Contractor shall deliver all tank material, tools, and appliances f. o. b. cars nearest railroad station. The Company shall haul all tank material, tools, and appliances necessary for the construction of the tank from the railroad station, and shall deliver same within 100 feet of the tank site, and on completion of the work shall return tools and appliances to the railroad station.

(3) *Drawings*.—The following drawings form an integral part of and are to be used in conjunction with these specifications: Plates II, III, IV, V, VI, VII, and VIII.

(4) *Dimensions*.—Diameter, 114 feet 6 inches, average inside measurement; height, 30 feet.

(5) *Material*.—All material for the tank shall be of steel, complying with Standard Specifications for Structural Steel of the American Society for Testing Materials (copy of which is hereto attached). Contractor to furnish a certificate from an approved firm of testing engineers covering all materials used, but such certificate shall not act to prevent Company exercising the right to reject undergaged plates or defective material wherever it may be found.

Tank to be composed of 6 rings of equal height, as shown on Plate II. Plates to be of uniform size with minimum dimensions, as hereinafter designated.

All plates shall be ordered to gage, permissible variations to be in accord with specifications for structural steel above referred to.

(6) *Punching and riveting, etc.*—Plates and angles for the shell of the tank must be rolled to proper curvature.

Plates and angles must be punched from the sides that are to be in contact, and the punching must be so accurate that the holes will match within 10 per cent of their diameter when plates are assembled. Holes for hot rivets shall be punched

^a 42-gallon barrels.

not more than one-sixteenth of an inch larger in diameter than the rivets that are fill them.

Riveting shall be done with pneumatic tools, an air pressure of approximately pounds per square inch at the receiver being used. Rivets must conform with specifications in diameter, length, pitch, and marginal distance. Should any burn or deformed, or loose rivets be found in the work, they are to be cut out and replaced. No calking of rivets will be allowed. Riveting of the bottom sheets shall be done from the inside; all other riveting, including riveting of the bottom to the bottom angle and of the roof, shall be done from the outside. All rivets one-half inch in diameter or larger shall be driven hot.

(7) *Calking*.—All edges to be calked shall be beveled by planing. All seams must be thoroughly calked with a round-nosed pneumatic calking tool. The shell of the tank shall be formed by ascending inside courses calked on the outside. Bottom plates to be calked on the inside except at angle iron, where they are to be calked from the outside, and both legs of angle iron shall be calked inside. Angle irons to butt calked at joints.

All roof plates to be calked from the outside of tank, this to include also calking top angle iron.

All castings, nozzles, flanges, and manheads riveted to the tank must be calked thoroughly inside and outside.

(8) *Testing the bottom*.—Upon completion of the bottom and the first ring of the shell and before the bottom has been lowered to the ground, the bottom shall be covered with water to a depth of not less than 5 inches, and all leaks that develop shall be made tight to the satisfaction of the Company or its duly authorized representative before any lowering is done. Necessary water for the testing shall be furnished by the Company for one test only. Should additional tests be made, water will be charged to the contractor at cost.

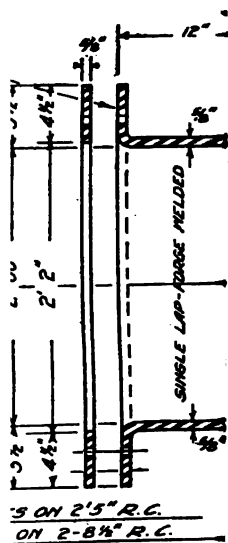
(9) *Thickness of metal, spacing of rivets, etc.*—The thickness of metal, the spacing of rivets, and the like, shall be as set forth in the following table:

Thickness of metal, spacing of rivets, etc., prescribed for tank covered by these specifications

Part.	Thick-ness.	Weight per square foot.	Horizontal riveting—all single rows.		Vertical riveting.			
			Diam-eter of rivets.	Pitch.	Rows.	Diam-eter of rivets.	Pitch.	Dist-ance between rows, center to center.
	Inches.	Pounds.	Inches.	Inches.		Inches.	Inches.	Inches.
Bottom sketch plates.....	$\frac{1}{2}$	12.75		13				
Bottom rectangular plates.....	$\frac{1}{2}$	10.20		13				
First ring.....	$\frac{1}{2}$	22.95	1	24	Triple.		3	
Second ring.....	$\frac{1}{2}$	20.40		24	do.		3	
Third ring.....	$\frac{1}{2}$	16.58		24	Double.		24	
Fourth ring.....	$\frac{1}{2}$	16.75		24	do.		24	
Fifth ring.....	$\frac{1}{2}$	10.20		2	do.		24	
Sixth ring.....	$\frac{1}{2}$	10.20		2	do.		24	
Roof plates.....	$\frac{1}{2}$	7.65		13				

(10) *Size of plates and angles*.—Plates for shell shall be 60 by 180 inches center to center of rivet laps (24 plates per ring). Bottom and top rectangular plates shall be 60 by 180 inches center to center of rivet laps. Bottom angle irons connecting the bottom and the shell shall be 4 by 4 inches by $\frac{1}{2}$ inch. Top angle iron connecting the roof with the shell shall be 3 by 3 inches by $\frac{1}{2}$ inch.

(11) *Angle shoes*.—Shoes uniting ends of angles shall be made of $\frac{1}{2}$ -inch 16-pound steel, and shall be not less than 12 inches in length with the ends drawn to a thin edge.



MANHOLE

2 WANTED - COM



Shoes must be set between steel angles, shell and bottom of tank, and must be of sufficient width to be properly calked outside the line of the steel angles.

(12) *Shell*.—The shell shall be composed of 6 courses, as shown on Plate II, each course to be a true circle and to be free from flaws and buckles. The first two courses, counting from the bottom upward, are to have the vertical seams triple riveted. The third, fourth, and fifth courses are to have the vertical seams double riveted. All horizontal seams are to be single riveted.

(13) *Roof*.—Roof to be composed of $\frac{1}{4}$ -inch steel plates weighing 7.65 pounds per square foot, and to rest on steel roof supports as shown on Plate III. The roof, however, shall not be riveted to roof supports at any point. When complete the roof must be "gas tight," and shall show no leaks when tested with an air pressure equal to the weight of the roof.

(14) *Roof supports*.—Roof supports shall be constructed of steel shapes as shown on Plate III. Size and fabrication to conform to those shown, and workmanship to be approved by the Company.

(15) *Manholes*.—Two manholes shall be placed in the first course of tank, as designated by the Company in the field. Each manhole shall be 20 inches deep, and shall be made of steel with welded seam, weighing not less than 21 pounds per square foot. Neck of flange must be covered to suit radius of tank shell. The manhole shall be fitted with steel cover plate five-eighths inch thick, faced and punched to suit holes in flange, and bolted to flange with 24 $\frac{1}{2}$ -inch square-head bolts and hexagon nuts. The manhole shall be riveted to tank shell with $\frac{1}{2}$ -inch rivets, as shown on Plate II.

(16) *Pipe flanges, nozzles, etc.*—The tank shall be furnished with the following pressed-steel flanges secured to the shell with $\frac{1}{2}$ -inch diameter rivets. Position to be designated in field by the Company. One pair of flanges to be threaded for 8-inch pipe for drawing off water. One pair of flanges to be threaded for 8-inch pipe for oil-inlet pipe. Bottom of tank to be fitted with one single flange threaded for 8-inch pipe, secured to bottom of tank with $\frac{1}{2}$ -inch rivets, location to be designated in the field by the Company. Roof of tank to be fitted with two 8-inch and one 4-inch pressed-steel flanges located in field by the Company, also with manholes and gage hatch, as shown in detail on Plate III.

Tank to be fitted with one combined swing pipe and suction nozzle, as shown in detail on Plate II. Nozzle to be made of single-lap forge-welded steel plate $\frac{1}{2}$ inch thick, and to be secured to shell of tank in position located in the field by the Company, with a double row of $\frac{1}{2}$ -inch rivets, as shown on drawing. Swing pipe and suction end of nozzle to be fitted with cast-steel flange, as shown.

(17) *Swing pipe*.—Contractor shall supply and install swing pipe as shown in detail on Plate IV. Swing pipe to be delivered on the job so that it may be placed (but not installed) inside of the tank shell as soon as the bottom is tested and lowered. Contractor will not be permitted to lift the swing pipe over the shell of the tank after more than the first ring of the tank is in position.

(18) *Double-swivel joint for swing pipe*.—Contractor shall supply and install to the satisfaction of the Company a double-swivel elbow joint for connecting the swing pipe to the swing-pipe nozzle. Details for this swivel joint are shown on Plate V. Construction of joint must follow closely details as per drawing.

(19) *Stairway*.—The Contractor shall furnish and install a stairway anchored to the shell of the tank and running from the ground to the top of the tank, as shown on Plate VI. This stairway shall be equipped with door covered with No. 12 wire screen, $\frac{1}{2}$ -inch mesh, as shown on Plate VI.

(20) *Explosion hatches*.—The tank shall be equipped with eight explosion hatches to be located on roof of tank in field as designated by the Company. Details of explosion hatches are shown on Plate VII. These explosion hatches are to be furnished by Contractor and must conform closely to drawing. Brass rods must be true and

smoothly turned, and must be accurately fitted in babbitted bearings, as shown, so that aluminum cover of hatch will rise without binding.

(21) *Cable guides and stuffing box.*—Tank shall be provided with cable guide stuffing box for cable, as shown in detail on Plate VIII. One hundred feet of diameter 6-strand 19-wire plow-steel galvanized-wire cable shall be furnished by Contractor. Sheave wheels for cable must be accurately centered and securely riveted to tank shell and calked inside and outside of tank.

(22) *Swing-pipe winch.*—Swing-pipe winch shall be furnished by Contractor as detail shown on Plate VIII, and shall be placed in position by Contractor designated by the Company.

(23) *Painting.*—The outside of the tank, including the top, shall be painted with one coat of asphaltic, graphite, or other paint approved by the Company, to be applied on with brush and to thoroughly cover the metal. The bottom of the tank shall be painted with three coats of the same paint, at least 6 hours being allowed for each successive coat to dry.

(24) *Inspection.*—All material and workmanship shall be subject at all times to inspection of the Company or its duly authorized representative, and any defective material, whether discovered before or after it has been used in the work, shall be removed and replaced in the construction by the Contractor at his own expense. Contractor shall also furnish transportation from railroad to tank site for such new material.

(25) *Final test.*—Upon the completion of the tank, and before it is inspected, the tank shall be filled with oil or water at the option and expense of the Company, and after that develop shall be made tight to the satisfaction of the Company by the Contractor at his own expense.

(26) *Boarding of work crew.*—The Contractor shall provide board, lodging, and transportation for his men at all times. Commissary water will be furnished by the Company. Purifying of same, where necessary, to be done by the Contractor. No eating liquors shall be permitted on the premises.

(27) *Workmanship.*—The work throughout shall be done in a first-class workmanlike manner, and only men competent in their line shall be employed about the tank. Upon demand of the Company, or its duly authorized representative, any workman who in the judgment of the Company is found to be incompetent, careless, or disobedient shall be dismissed.

(28) *Extra work.*—These plans and specifications are intended to describe a complete oil-tight and gas-tight tank. Any work and material necessary to produce a tank, although not mentioned in the specifications or shown on the plans, shall be supplied by the Contractor without extra cost to the Company. The Company shall not pay for extra work unless it has been executed on a written order by the Company or its duly authorized representative.

(29) *Rubbish.*—On completion of the work all useless material used in the construction shall be cleared away from inside and outside the tank, and shall be removed to such a point upon the premises as may be designated by the Company.

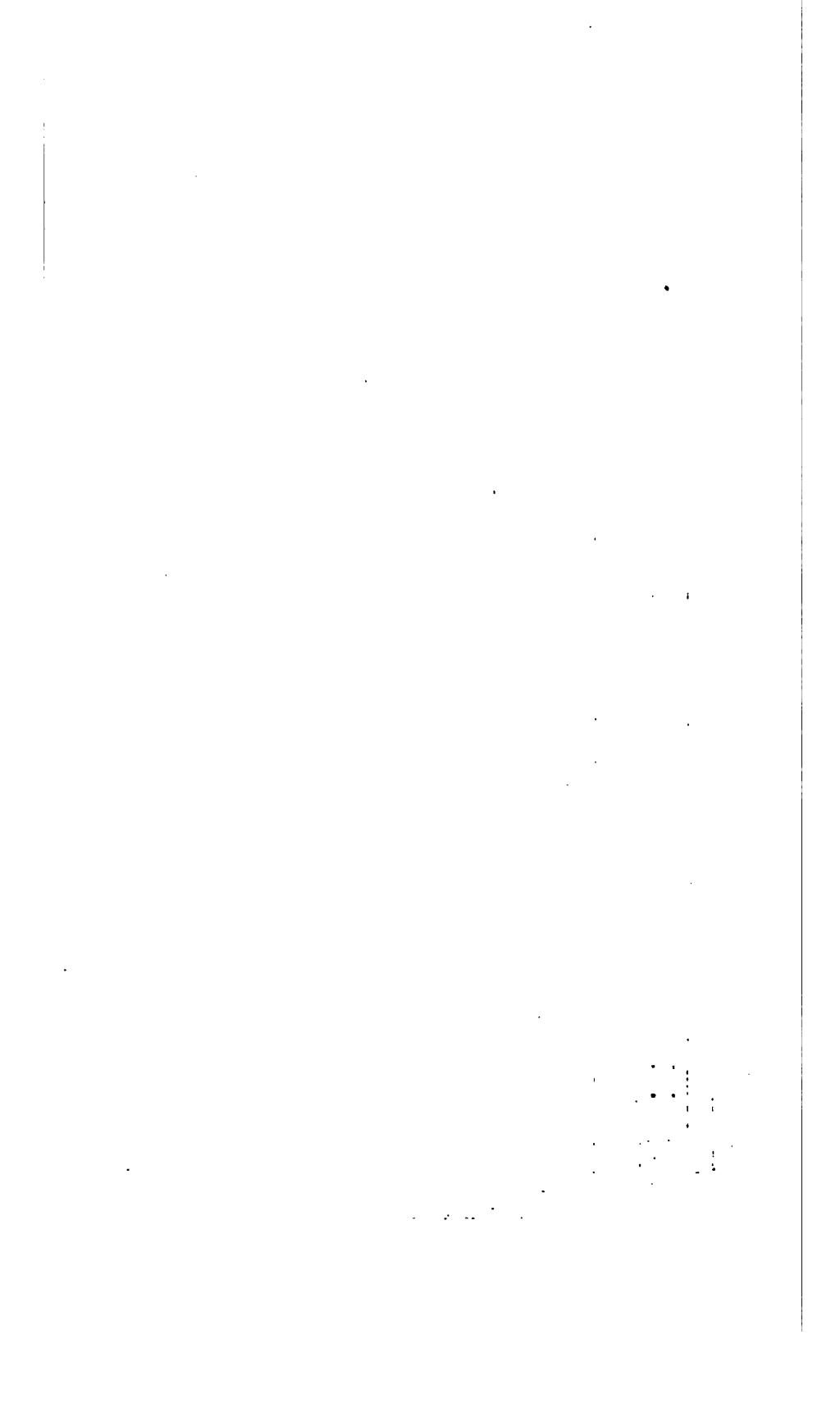
(30) *Camp sanitation.*—The Contractor shall at all times maintain his boiler house, sleeping quarters, and all other appurtenant facilities in a sanitary condition to the satisfaction of the Company.

1/8" L
1/2" L
LAT BARS

1/8" L
1/2" L

1/8" L
1/2" L

3
100-BAR



AMERICAN SOCIETY FOR TESTING MATERIALS.

Philadelphia, Pa., U. S. A.

Affiliated with the
International Association for Testing Materials.

STANDARD SPECIFICATIONS

FOR

STRUCTURAL STEEL FOR BUILDINGS.

Serial designation: A 9-16.

Adopted 1901; revised 1909, 1913, 1914, 1916.

I. MANUFACTURE.

Process.

1. (a) Structural steel, except as noted in paragraph b, may be made by the Bessemer or the open-hearth process.

(b) Rivet steel, and steel for plates or angles over $\frac{1}{4}$ inch in thickness which are to be punched, shall be made by the open-hearth process.

II. CHEMICAL PROPERTIES AND TESTS.

Chemical composition.

2. The steel shall conform to the following requirements as to chemical composition:

	Structural steel.	Rivet steel.
Phosphorus (Bessemer.....)	Not over 0.10 per cent.	
(Open-hearth.....)	Not over 0.06 per cent.	Not over 0.06 per cent.
Sulphur.....		Not over 0.045 per cent.

Ladle analyses.

3. An analysis of each melt of steel shall be made by the manufacturer to determine the percentages of carbon, manganese, phosphorus, and sulphur. This analysis shall be made from a test ingot taken during the pouring of the melt. The chemical composition thus determined shall be reported to the purchaser or his representative, and shall conform to the requirements specified in section 2.

Check analyses.

4. Analyses may be made by the purchaser from finished material representing each melt. The phosphorus and sulphur content thus determined shall not exceed that specified in section 2 by more than 25 per cent.

III. PHYSICAL PROPERTIES AND TESTS.

Tension tests.

5. (a) The material shall conform to the following requirements as to tensile properties:

Properties considered.	Structural steel.	Rivet steel.
Tensile strength, pounds per square inch.....	55,000 to 65,000.....	46,000 to 56,000.
Yield point, minimum, pounds per square inch.	0.5 tensile strength.....	0.5 tensile strength.
Elongation in 8 inches, minimum, per cent. . .	1,400,000	1,400,000
Elongation in 2 inches, minimum, per cent. . .	Tensile strength ^a	Tensile strength
	22.....	

^a See section 6.

(b) The yield point shall be determined by the drop of the beam of the testing machine.

Modifications in elongation.

6. (a) For structural steel over $\frac{3}{4}$ inch in thickness, a deduction of 1 from the percentage of elongation in 8 inches specified in section 5, a, shall be made for increase of $\frac{1}{4}$ inch in thickness above $\frac{3}{4}$ inch, to a minimum of 18 per cent.

(b) For structural steel under $\frac{1}{4}$ inch in thickness, a deduction of 2.5 from the percentage of elongation in 8 inches specified in section 5, a, shall be made for decrease of $\frac{1}{4}$ inch in thickness below $\frac{1}{4}$ inch.

Bend tests.

7. (a) The test specimen for plates, shapes, and bars, except as specified in paragraphs b and c, shall bend cold through 180 degrees without cracking on the outside of the bent portion, as follows: For material $\frac{3}{4}$ inch or under in thickness, flat or around a pin the diameter of which is equal to the thickness of the specimen; and for material over $\frac{3}{4}$ inch to and including $1\frac{1}{2}$ inches in thickness, around a pin the diameter of which is equal to the thickness of the specimen; and for material over $1\frac{1}{2}$ inches in thickness, around a pin the diameter of which is equal to twice the thickness of the specimen.

(b) The test specimen for pins, rollers, and other bars, when prepared as specified in section 8, e, shall bend cold through 180 degrees around a 1-inch pin without cracking on the outside of the bent portion.

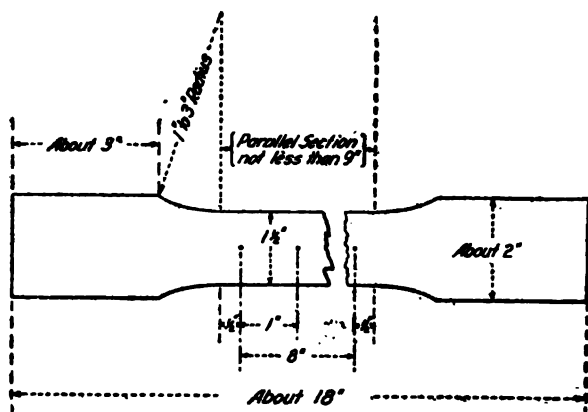


FIGURE 1.—Form and dimensions of test specimen for tension and bending tests of plates, shapes, and bars.

(c) The test specimen for rivet steel shall bend cold through 180 degrees flat without cracking on the outside of the bent portion.

Test specimens.

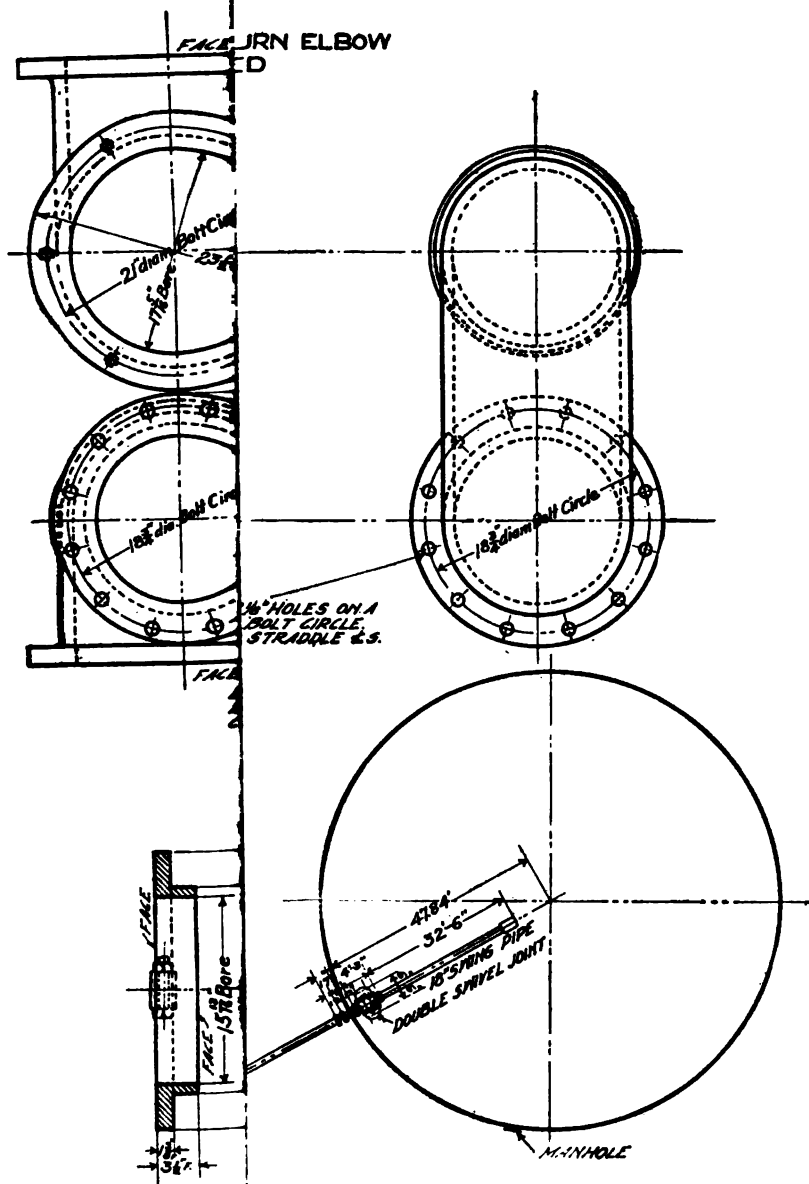
8. (a) Specimens for tension and bending tests shall be taken from rolled material in the condition in which it comes from the rolls, except as specified in paragraph (b).

(b) Specimens for tension and bending tests of pins and rollers shall be taken from the finished bars, after annealing when annealing is specified.

(c) Specimens for tension and bending tests of plates, shapes, and bars, except as specified in paragraphs d, e, and f, shall be of the full thickness of material as received and may be machined to the form and dimensions shown in figure 1, or may be of any other form parallel.

(d) Specimens for tension and bending tests of plates over $1\frac{1}{2}$ inches in thickness may be machined to a thickness or diameter of at least $\frac{3}{4}$ inch for a length of 9 inches.

(e) Specimens for tension tests of pins, rollers, and bars more than $1\frac{1}{2}$ inches in thickness or diameter may conform to the dimensions shown in figure 2. In this case the ends shall be of a form to fit the holders of the testing machine in such a way that



load shall be axial. Bend test specimens may be 1 by $\frac{1}{4}$ inch in section. The axis of the specimen shall be located at any point midway between the center and surface and shall be parallel to the axis of the bar.

(f) Specimens for tension and bending tests of rivet steel shall be of the full-size section of bars as rolled.

Number of tests.

9. (a) One tension and one bending test shall be made of material from each melt; except that if material from one melt differs $\frac{1}{8}$ inch or more in thickness, one tension and one bending test shall be made from both the thickest and the thinnest material rolled.

(b) If any test specimen shows defective machining or develops flaws, it may be discarded and another specimen substituted.

(c) If the percentage of elongation of any tension-test specimen is less than that specified in section 5, a, and any part of the fracture is more than $\frac{1}{4}$ inch from the center of the gage length of a 2-inch specimen or is outside the middle third of the gage length of an 8-inch specimen, as indicated by scribe scratches marked on the specimen before testing, a retest shall be allowed.

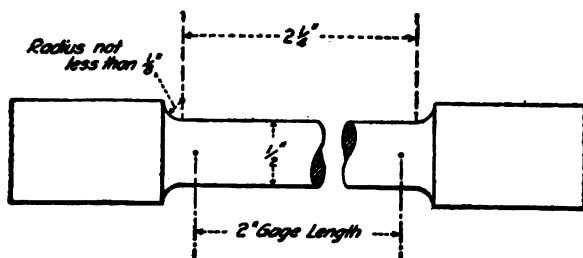


FIGURE 2.—Form and dimensions of test specimen for tension tests of pins, rollers, and bars more than $\frac{1}{4}$ inches thick.

IV. PERMISSIBLE VARIATIONS IN WEIGHT AND THICKNESS.

Permissible variations.

10. The cross section or weight of each piece of steel shall not vary more than 2.5 per cent from that specified; except in the case of sheared plates, which shall be covered by the following permissible variations. One cubic inch of rolled steel is assumed to weigh 0.2833 pound.

(a) When ordered to weight per square foot: The weight of each lot^a in each shipment shall not vary from the weight ordered more than the amount given in Table 1.

(b) When ordered to thickness: The thickness of each plate shall not vary more than 0.01 inch under that ordered.

The overweight of each lot^b in each shipment shall not exceed the amount given in Table 2.

^a The term "lot" applied to Table 1 means all of the plates of each group width and group weight.

^b The term "lot" applied to Table 2 means all of the plates of each group width and group thickness.

TABLE 1.—Permissible variations of plates ordered to weight.

Ordered weight.	Permissible variations in average weights per square foot of plates for widths given. (Expressed in percentages of ordered weights.)																Ordered weight.		
	Under 48 inches.		48 to 60 inches, exclusive.		60 to 72 inches, exclusive.		72 to 84 inches, exclusive.		84 to 96 inches, exclusive.		96 to 108 inches, exclusive.		108 to 120 inches, exclusive.		120 to 132 inches, exclusive.			132 inches or over.	
	Over.	Under.	Over.	Under.	Over.	Under.	Over.	Under.	Over.	Under.	Over.	Under.	Over.	Under.	Over.	Under.		Over.	Under.
<i>Pounds per square foot.</i>	5	3	5.5	3	6	3	7	3	6	3	7	3	8	3	9	3	10	Under 5.	
Under 5	4.5	3	5	3	5.5	3	6	3	5	3	6	3	7	3	8	3	9	5 to 7.5, exclusive.	
5 to 7.5, exclusive.	4	3	4.5	3	5	3	5.5	3	5	3	6	3	7	3	8	3	9	7.5 to 10, exclusive.	
7.5 to 10, exclusive.	3.5	2.5	4	3	4.5	3	5	3	4.5	3	5.5	3	6	3	7	3	8	10 to 12.5, exclusive.	
10 to 12.5, exclusive.	3	2.5	3.5	3	4	3	4.5	3	4	3	5	3	6	3	7	3	8	12.5 to 15, exclusive.	
12.5 to 15, exclusive.	2.5	2.5	3	2.5	3.5	3	4	3	3.5	3	4.5	3	5	3	6	3	7	15 to 17.5, exclusive.	
15 to 17.5, exclusive.	2.5	2.5	2.5	2.5	3	2.5	3.5	2.5	3.5	2.5	4.5	3	5.5	3	6	3	7	17.5 to 20, exclusive.	
17.5 to 20, exclusive.	2.5	2	2.5	2.5	3	2.5	3.5	2.5	3.5	2.5	4.5	3	5.5	3	6	3	7	20 to 25, exclusive.	
20 to 25, exclusive.	2.5	2	2	2	2.5	2.5	3	2.5	3	2.5	4	3	5	3	6	3	7	25 to 30, exclusive.	
25 to 30, exclusive.	2	2	2	2	2	2	2.5	2.5	2.5	2.5	3.5	3	4.5	3	5.5	3	6	30 to 40, exclusive.	
30 to 40, exclusive.	2	2	2	2	2	2	2	2	2	2	2.5	2.5	3	3	4	3	5	40 or over.	
40 or over.	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	40 or over.	

NOTE.—The weight per square foot of individual plates shall not vary from the ordered weight by more than $\frac{1}{4}$ times the amount given in this table.

TABLE 2.—Permissible overweights of plates ordered to thickness.

Ordered thickness.	Permissible excess in average weights per square foot of plates for widths given. (Expressed in percentages of nominal weights.)								Ordered thickness.
	Under 48 inches.	48 to 60 inches, exclusive.	60 to 72 inches, exclusive.	72 to 84 inches, exclusive.	84 to 96 inches, exclusive.	96 to 108 inches, exclusive.	108 to 120 inches, exclusive.	120 to 132 inches, exclusive.	132 inches or over.
<i>Inches.</i>									
Under 1.....	9	10	12	14	12	12	14	16	Under 1.....
1 to 1 $\frac{1}{2}$, exclusive.....	8	9	10	12	10	10	12	14	1 to 1 $\frac{1}{2}$, exclusive.....
1 $\frac{1}{2}$ to 2, exclusive.....	7	8	9	10	9	9	10	12	1 $\frac{1}{2}$ to 2, exclusive.....
2 to 2 $\frac{1}{2}$, exclusive.....	6	7	8	9	8	8	9	10	2 to 2 $\frac{1}{2}$, exclusive.....
2 $\frac{1}{2}$ to 3, exclusive.....	5	6	7	8	7	7	8	9	2 $\frac{1}{2}$ to 3, exclusive.....
3 to 3 $\frac{1}{2}$, exclusive.....	4.5	5	6	7	6	6	7	8	3 to 3 $\frac{1}{2}$, exclusive.....
3 $\frac{1}{2}$ to 4, exclusive.....	4	4.5	5	6	5	5	6	7	3 $\frac{1}{2}$ to 4, exclusive.....
4 to 4 $\frac{1}{2}$, exclusive.....	3.5	4	4.5	5	4	4	5	6	4 to 4 $\frac{1}{2}$, exclusive.....
4 $\frac{1}{2}$ to 5, exclusive.....	3	3.5	4	4.5	3.5	3.5	4	5	4 $\frac{1}{2}$ to 5, exclusive.....
5 to 6, exclusive.....	2.5	3	3.5	4	3	3	4	5	5 to 6, exclusive.....
6 to 8, exclusive.....	2.5	2.5	3	3.5	2.5	2.5	3	4	6 to 8, exclusive.....
8 to 10, exclusive.....	2.5	2.5	3	3.5	2.5	2.5	3	4	8 to 10, exclusive.....
10 or over.....	2.5	2.5	3	3.5	2.5	2.5	3	4	10 or over.....

V. FINISH.

Finish.

11. The finished material shall be free from injurious defects and shall be of workmanlike finish.

VI. MARKING.

Marking.

12. The name or brand of the manufacturer and the melt number shall be stamped or rolled on all finished material, except that rivet and lattice bars and small sections shall, when loaded for shipment, be properly separated and marked for identification. The identification marks shall be legibly stamped on the end of each pin and roller. The melt number shall be legibly marked, by stamping, if practicable, on each test specimen.

VII. INSPECTION AND REJECTION.

13. The inspector representing the purchaser shall have free entry, at all times, while work on the contract of the purchaser is being performed, to all parts of the manufacturer's works that concern the manufacture of the material ordered. The manufacturer shall afford the inspector, free of cost, all reasonable facilities to enable him that the material is being furnished in accordance with these specifications. All tests (except check analyses) and inspection shall be made at the place of manufacture prior to shipment, unless otherwise specified, and shall be so conducted as not to interfere unnecessarily with the operation of the works.

14. (a) Unless otherwise specified, any rejection based on tests made in accordance with section 4 shall be reported within five working days from the receipt of analyses.

(b) Material that shows injurious defects subsequent to its acceptance at the manufacturer's works will be rejected, and the manufacturer will be notified.

Rehearing.

15. Samples tested in accordance with section 4, which represent rejected material, shall be preserved for two weeks from the date of the test report. In case of dissatisfaction with the results of the tests, the manufacturer may make claim for a rehearing within that time.

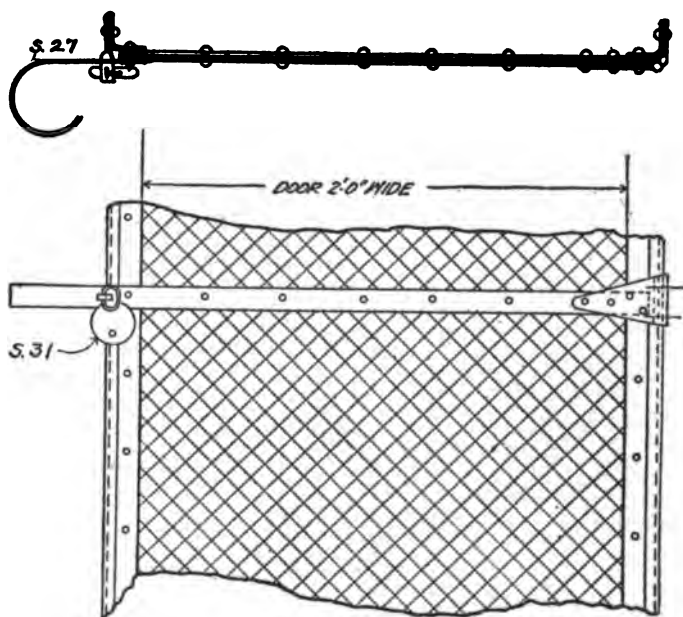
DISCUSSION OF TANK SPECIFICATIONS.

SAND-LINE CONSTRUCTION.

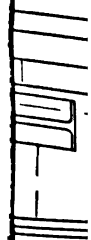
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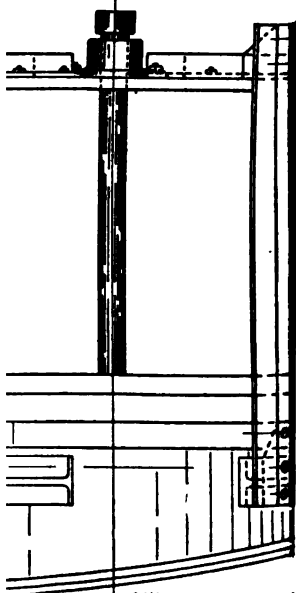
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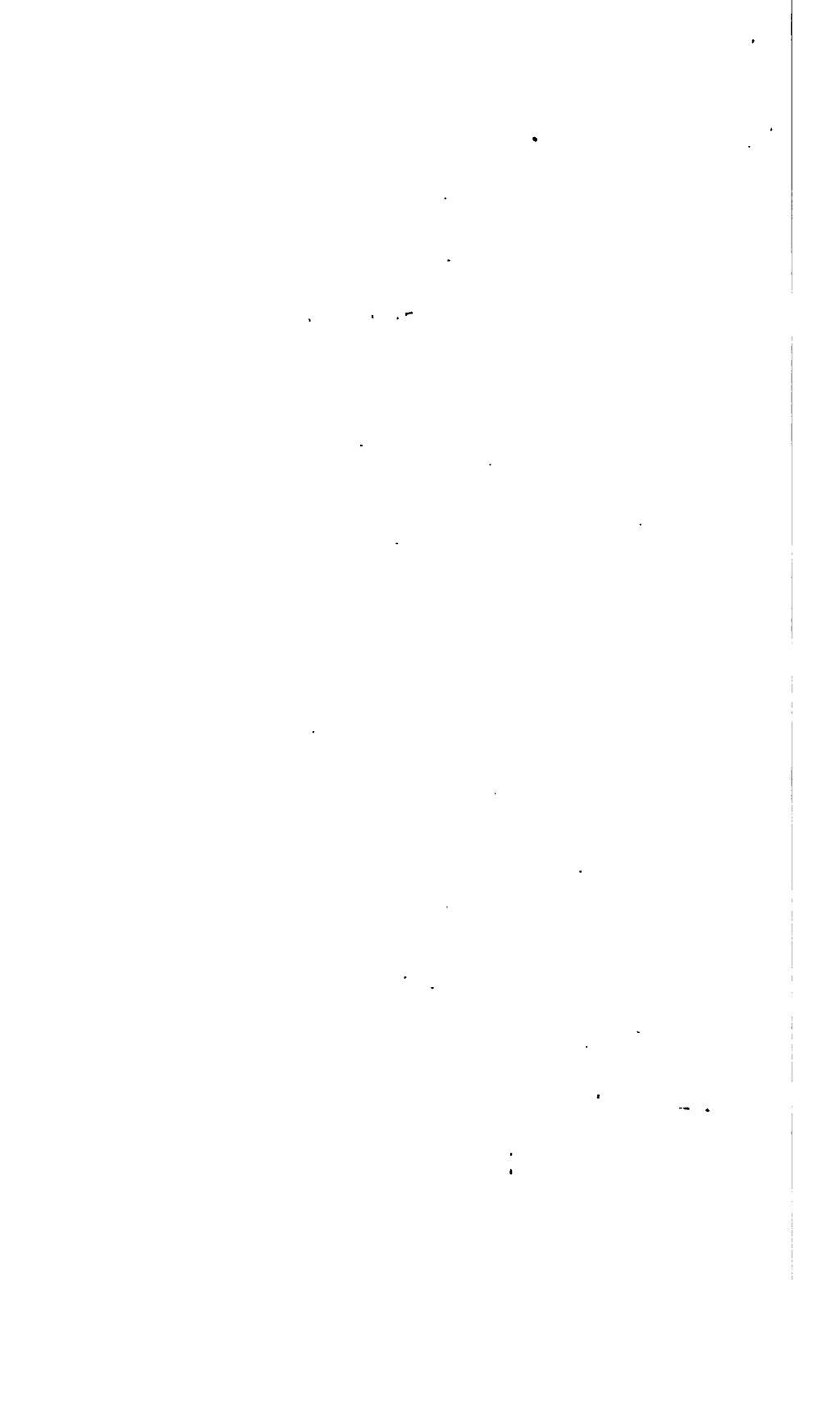
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This method of wind bracing is of reasonable cost, is effective, and is easily applied to any tank. It consists of stringing $\frac{3}{4}$ -inch flexible steel cables across 4 diameters of the tank, and at such distances apart that the tank area is divided into 45° segments, giving the appearance, in horizontal projection, of spokes in a gigantic wheel. Two sets of cables are generally used—one in a plane about 10 feet above the bottom of the tank, and the other 20 feet above the bottom, the "spokes" in the two planes being staggered so that the shell in a 55,000-barrel tank is wind-braced at intervals of approximately 20 feet along its periphery. Some engineers place 3 sets of cables, the topmost being attached to the top angle iron. This precaution may well be used for wooden-roof tanks with wooden roof supports, but should not be necessary when steel construction is used throughout.

Formerly old sand lines that had been discarded from drilling rigs were used—hence the name. However, for obvious reasons, the use of such lines is not good practice. Cables about a drilling rig are seldom discarded until they have become badly worn and their tensile strength consequently so much decreased that there is danger that the line will break under any heavy strain, with disastrous results.

The lines are fastened to the shell of the tank by small angle irons riveted to it, or by through-going eyebolts each of which is provided with a nut, washer, and gasket on either side of the plate. At the center of the tank the lines are attached to a heavy steel ring surrounding the center post, and each radius or "spoke" is provided with a turnbuckle, so that a practically uniform tension may be obtained on all lines, and whatever adjustment from time to time is necessary to keep this tension constant can be made. As the center ring can easily be constructed in two sections, this type of wind bracing can readily be applied to completed tanks as well as to those in process of erection. The only precaution necessary is to see that the diameters upon which the lines are strung are chosen so as to clear the roof supports.

TANK ROOFS.

The tank roof specified is composed of material of sufficient thickness to permit calking. Some companies prefer to use a lighter plate and to get the necessary tightness at the seams by using a thread "weave" immersed in red lead and placed between the laps of the sheets before they are riveted together. This method cuts down the cost of construction by decreasing the amount of metal used, but is criticized by many on the ground that the light hydrocarbon gases rising from the oil will in time "cut" the saturating material of the thread "weave" and completely destroy the tightness of the joint. A further objection put forth is that the roof of a tank is the part

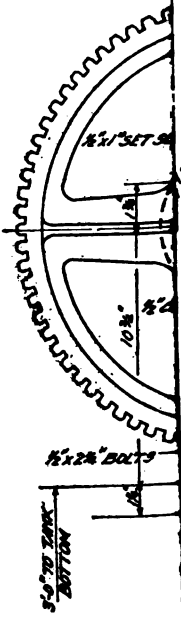
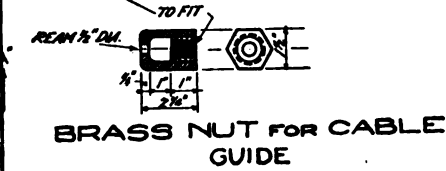
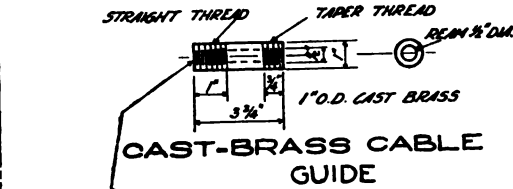
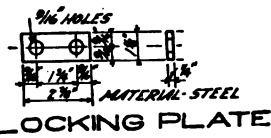
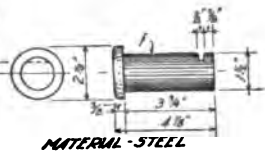
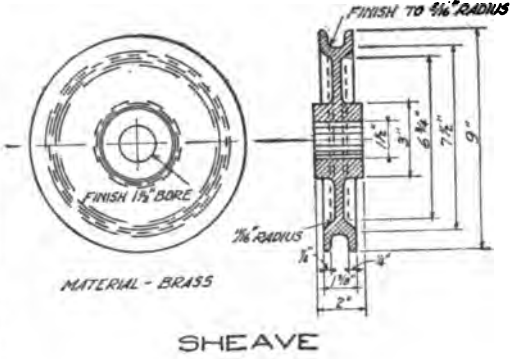
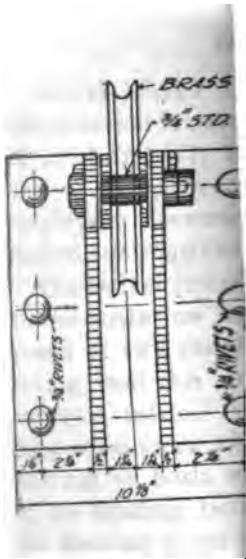
that deteriorates most rapidly owing to the action of these same hydrocarbon gases, especially if the oil contains any appreciable amount of sulphur, and a plate thinner than that which can be successfully calked (three-sixteenths of an inch) is undesirable. On the other hand, the objection can be raised to the use of the heavier material not only that it increases the cost of the tank, but that a roof is continuously subject to expansion and contraction stresses as well as to atmospheric pressures which cause it to "breathe." Thus the seams are subjected to distortion and it is questionable whether a calked joint can be kept tight.

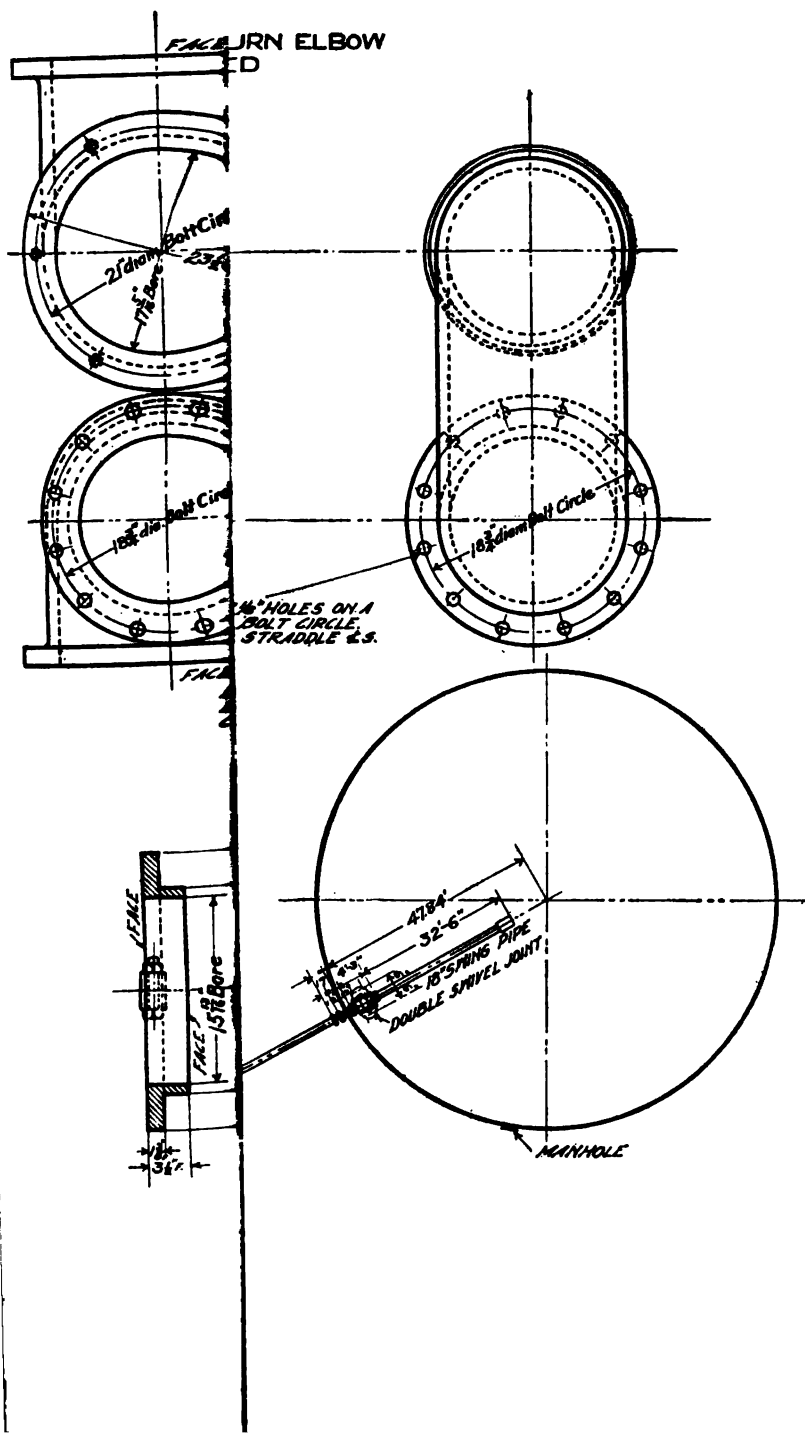
SIZE OF PLATES.

The size of plate ordinarily used for the shell of the tank is about 5 feet wide by 15 feet long (24 sheets to the ring). Six of such rings are required to obtain the necessary height of tank. Some companies, however, notably the Shell Oil Co. of California, are using sheets of sufficient width to require only 4 rings for a 30-foot tank. They are approximately 7 feet 6 inches wide and 18 feet long. By using plates of such size it is possible to eliminate two horizontal seams and four vertical seams, thus lessening the length of calked joints in the shell by more than 15 per cent. On the other hand, the amount of metal in the shell of a 4-ring tank is necessarily more than in a 6-ring tank because of the thickness of material required to satisfy proper engineering design and the impracticability of rolling a taper sheet. It is argued that if any extra thickness of metal is to be used, it should be put into the top or the bottom plates, as these are subject to the greatest deterioration by corrosive action, and not into the vertical shell. Further, the laps of sheets when properly riveted and calked seldom leak, and if leaks do occur, they can be easily repaired because of their accessibility.

SWING PIPES.

The swing pipes used by the various companies are all of the same general type, differing only as to details of construction. They range in size from 6 to 24 inches in diameter, depending upon the gravity of the oil to be handled and the size of suction line that they feed. Some operators equip their swing pipes with floats, making it possible to draw the oil from a point always at a certain definite distance below the surface by merely "slacking up" on the cable sufficiently to allow the float to carry the full weight of the pipe. There is the disadvantage, however, that should it be necessary at any time to raise the swing pipe when the tank is empty, the float makes the pipe unwieldy to handle, and if the pipe is made up of thin sheets of riveted plate, the usual practice, the resulting strain may produce leaky joints.





V. FINISH.

Finish.

11. The finished material shall be free from injurious defects and shall have workmanlike finish.

VI. MARKING.

Marking.

12. The name or brand of the manufacturer and the melt number shall be legibly stamped or rolled on all finished material, except that rivet and lattice bars and small sections shall, when loaded for shipment, be properly separated and marked for identification. The identification marks shall be legibly stamped on the end of each pin and roller. The melt number shall be legibly marked, by stamping practicable, on each test specimen.

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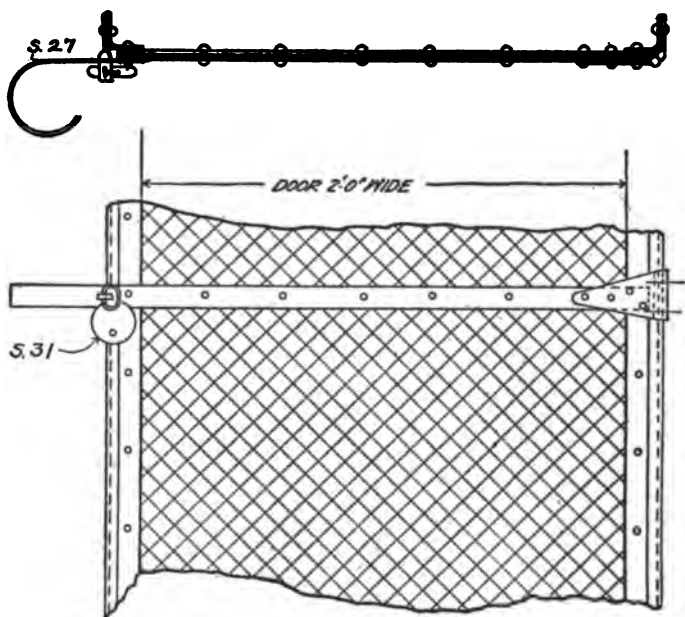
DISCUSSION OF TANK SPECIFICATIONS.

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diameter of 488 feet, a total depth of about 25 feet, and a slope on the sides of 1 to 1. The roof is constructed of wood covered with roofing paper. The specifications follow.

SPECIFICATIONS FOR CONSTRUCTION OF CONCRETE-LINED OIL-STORAGE RESERVOIR.

SECTION I.—GENERAL REQUIREMENTS.

(1) DEFINITION.

The ——— Company in itself, or by its duly authorized representative, shall be referred to in these specifications as the "Company."

The firm undertaking to furnish the material and erect the reservoir as herein specified, in itself, or by its duly authorized representative, shall be referred to in these specifications as the "Contractor."

At all times during the progress of the work the Contractor shall designate some person as his representative to whom instructions may be given by duly authorized representatives of the Company.

(2) LOCATION.

The reservoir shall be located on the land of the Company in sec. —, T. —, R. —.

(3) DRAWINGS.

The drawings listed below, showing the details of the construction of the reservoir, are hereby made a part of these specifications: Plates XI, XII, XIII, XIV, and XV.

The drawings and specifications are intended to supplement each other, so that the work described in the drawings and not mentioned in the specifications, or vice versa, is to be executed by the Contractor as if it were both mentioned in the specifications and shown on the drawings.

(4) SHAPE AND PLAN OF CONSTRUCTION.

The reservoir shall be circular in plan, as shown on Plate XI, and shall be constructed by making an excavation and constructing around the excavation, with the excavated material, an earthen embankment. The area within the inner crest of the embankment shall then be covered with a wooden roof, after which the bottom and the sides of the place inclosed shall be lined with concrete.

(5) DIMENSIONS, SLOPES, AREAS, AND CAPACITY OF EMBANKMENTS.

The dimensions of the reservoir are as follows:

Inside diameter at top, 488 feet.

Inside diameter at bottom, 437 feet 6 inches.

Maximum depth, approximately 25 feet 11 inches.

The slopes of the embankment shall be as follows:

Slope of embankment inside reservoir, 1 horizontal to 1 vertical.

Slope of embankment outside reservoir, 2 horizontal to 1 vertical.

Slope of embankment top of reservoir, 24 horizontal to 1 vertical.

The width of the embankment on top shall be 15 feet. The area of the bottom will be approximately 150,330 square feet. The area of the sides will be 53,200 square feet; total area, 203,530 square feet; capacity, in barrels of 42 gallons each, approximately 750,000.

(6) WORKMANSHIP.

All work in connection with the construction of the reservoir must be done in a neat workmanlike manner, and to the entire satisfaction of the Company, or its duly authorized representative.

(7) INSPECTION OF MATERIAL.

All material furnished by the Contractor as herein specified shall be of the best of its respective kind, and shall at all times be subject to inspection by the Company or its duly authorized representative, and any material that shall be found defective

and be condemned by the Company must be removed immediately by the Contractor and replaced by acceptable material.

(8) DAMAGES.

The Contractor shall at all times be responsible for damages to material that stored in the vicinity of the reservoir site during the process of its construction, and the Contractor shall make good such damaged material by supplying new material from time to time at his own cost and expense as directed by the Company. The Contractor shall also be responsible for any damages done to the reservoir, or to a part of it, or to any property of the Company upon which the reservoir is situated caused by the carelessness or negligence of any of his employees. The Contractor shall also be liable for all damages to tools, implements, and equipment furnished by him during the progress of the work.

(9) CARE OF MEN.

The Contractor shall have complete responsibility for the care of the men in his employ, and shall be liable for all damages by accident to such employees. He shall furnish the necessary commissary for feeding his men, and shall provide necessary tents, temporary houses, and equipment for sleeping quarters.

(10) INCOMPETENT EMPLOYEES.

The Contractor shall furnish only such men for the prosecution of the work as are competent in their particular line of employment. If any employee is judged incompetent by the Company or its duly authorized representative then the Contractor shall immediately remove such employee and shall not again employ him upon the work.

SECTION II.—EARTHWORK.

(1) NATURE OF THE WORK.

The work to be done under this section consists of excavating the necessary earth from the interior of the reservoir and depositing such excavated material in the embankments surrounding the reservoir, in accordance with plans hereto attached.

(2) EQUIPMENT.

The Contractor shall supply all the necessary labor, tools, and equipment, including picks, shovels, drills, "fresno" scrapers, wheel scrapers, road machines, tampers, and teams, and whatever equipment is necessary for properly carrying on the work.

(3) GRADE STAKES.

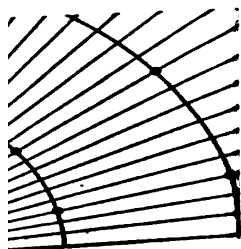
The Company shall place grade stakes wherever required about the reservoir, and the completed excavations and fills shall conform truly to the slope and outline determined by such grade stakes, and the Company shall not be responsible for not doing so. It shall it pay for any work that does not conform to such grade stakes. The Contractor and his employees shall at all times exercise due caution in caring for grade stakes as set by the Company, and any employee willfully destroying such stakes shall be immediately discharged by the Contractor.

(4) EXCAVATION.

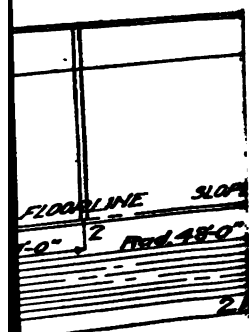
The exact amount of excavation shall be governed by the material that must be excavated, and shall be determined by the Company from time to time during the progress of the work. Should it be necessary in the opinion of the Company at any time to excavate more material from within the reservoir than originally intended, or to excavate material from borrow pits, the exact amount of such excavation shall be carefully determined by the Company by cross section, and the additional yardage shall be paid for at the same rate as the yardage from the original excavation.

(5) STRIPPING THE SURFACE SOIL.

The Contractor shall first strip the entire site of the reservoir, both that portion in cut and that portion to be covered by embankment, of all vegetable matter and top soil. This stripping shall extend to a depth of at least 8 inches below the surface and to whatever greater depth may in the judgment of the Company be required. The



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material so stripped shall be removed from the area and placed in fire levees surrounding the reservoir site, or at other points in the vicinity of the reservoir, as directed by the Company.

(6) OIL AND WATER DRAINS.

Two 12-inch oil lines and one 8-inch water drain shall be placed beneath the embankment, as shown on Plate XII, at such positions as shall be designated by the Company.

Ditches for these pipes shall be dug by the Contractor immediately after the surface soil has been stripped. Pipe shall be furnished and laid by the Company. The Contractor, however, shall do the work of backfilling the ditches, also of placing the concrete blocks, as shown on Plate XII.

(7) BUILDING UP THE EMBANKMENT.

The area beneath the embankment shall be plowed by the Contractor, thoroughly harrowed, and moistened with water to the satisfaction of the Company. The embankment shall then be built with material excavated from within the reservoir, or, if need be, from pits in the vicinity of the reservoir. The material may be excavated and placed in the embankment by wheel scrapers, "fresno" scrapers, or wagons. It shall, however, not be deposited in layers more than 3 inches thick, and where necessary road machines shall be employed to insure the proper spreading of the material in the embankment. Under no conditions shall material be dumped onto the embankment in a pile and not spread.

All material placed in the embankment shall be moistened with water before being loaded into the scrapers or wagons, and shall again be moistened after having been spread. Any excavated material from within the reservoir that, in the judgment of the Company, is not fit to be placed in the embankment shall be placed by the Contractor at a point outside the reservoir site, as designated by the Company.

When the material has been placed in thin layers in the embankment and moistened as specified, it shall then be tamped by at least two roller tampers of the sheep's foot type. Each tamper shall be drawn by a 4-horse team, and shall be driven around the top of the fill, making at least 10 complete circuits of the reservoir per hour.

Water for wetting material shall be furnished by the Company, and shall be brought by the Company in water mains surrounding the outer circumference of the reservoir and at a distance of about 20 feet from the outside toe of the embankment. The Contractor shall furnish and shall lay all necessary laterals from this water main into the reservoir site, and shall move such laterals from time to time as may be required by the progress of the work.

(8) BORROW PITS.

Where necessary, borrow pits shall be designated by the Company in the vicinity of the reservoir site. The nearest rim of such a borrow pit shall be at a distance not less than 200 feet from the outer toe of the reservoir embankment, and the farthest rim of such a borrow pit shall be at a distance not to exceed 400 feet from the toe of the embankment. If it is necessary to so locate a borrow pit that its nearest rim is a greater distance than 600 feet from the outer toe of the reservoir embankment, the Contractor shall receive extra compensation for the removal of such material as shall be determined on at the time.

(9) RUNWAYS TO EMBANKMENT.

In the construction of the embankment the Contractor shall build runways for the removal of the material from the excavation inside the reservoir to the embankment. These runways shall be located by the Company, and shall be spaced at a distance not less than 100 feet apart. The scrapers, wagons, teams, etc., in carrying the dirt from the excavation to the embankment shall be driven up one runway and shall return to the excavation along the top of the embankment and down the next adjacent runway.

(10) THE INSIDE SLOPE.

To obtain a homogenous and compact surface to the inner slope of the embankment, the inner slope shall be built and finished in the following manner. The embankment of the reservoir shall first be built up as above provided in these specifications. The excavation shall be so made that the inner slope shall be about the width of a wheel scraper beyond the line of the desired finished slope, beginning at the bottom of the reservoir which shall be excavated for a distance of about one foot below the position of the concrete lining. The inner slope shall then be built up by material excavated from within the reservoir, deposited by wheel scrapers in thin horizontal layers, and shall be moistened and thoroughly tamped by the sheep's foot tampers. This inner embankment, or earthen lining, shall be bound to the original embankment and slope by first plowing or otherwise roughening the original slope and embankment. When this lining is thus built up it shall be allowed to stand for at least 3 days before being trimmed to grade. The inner face of the embankment so built up shall then be trimmed to a true and even slope to conform with the dimensions of the reservoir as herein set forth.

(11) TOP OF THE EMBANKMENT.

The top of the embankment shall be built at least 2 inches higher than the finished top, and shall be made smooth and even and brought to proper grade.

(12) OUTSIDE SLOPE.

As herein specified, the outside slope of the embankment shall be 2 horizontal to 1 vertical. It shall be built up compactly, and neatly finished, and upon completion shall be oiled by the Contractor for a distance of 3 feet beyond the toe of the embankment. Oil shall be furnished by the Company on the ground and the Contractor shall provide the necessary equipment for sprinkling the oil upon the embankment.

(13) BOTTOM OF RESERVOIR.

The bottom of the reservoir shall be excavated to a subgrade 1 foot below the finished grade. It shall then be thoroughly plowed, harrowed, moistened, and compacted. Suitable refilling material shall then be spread over it to bring it up to the finished grade. The refilling material shall be spread on in thin layers as specified for the embankment, and shall be thoroughly moistened and rolled. The roller used on this work shall have a smooth surface, and shall weigh no less than 5 tons.

(14) EXCAVATION FOR POST FOOTINGS AND SWING-PIPE PIT.

When the bottom of the reservoir has been brought to the proper grade, the Contractor shall make excavations for the concrete footings for the posts, for the swing-pipe pit, and for the joint between the floor and the side slabs as designated by the Company. The Contractor shall remove from the inside of the reservoir all such material excavated, and shall place it at positions designated by the Company or its duly authorized representative.

(15) EXTRA WORK.

Extra work unless otherwise stated shall be done by force account, but under no conditions shall extra work be performed by the Contractor, or paid for by the Company, without written authority from the Company.

SECTION III.—ROOF.**(1) NATURE OF THE WORK.**

The work to be done, under this section, consists in the erection of a wooden roof over the reservoir as herein specified, and in accordance with drawings hereto attached.

(2) MATERIAL AND EQUIPMENT.

The Contractor shall supply all the necessary material, equipment, and labor for the construction of the roof. All such material shall be the best of its respective kind, and shall be subject at all times to the inspection of the Company or its duly authorized representative. It shall conform to the sizes as designated on the drawings.

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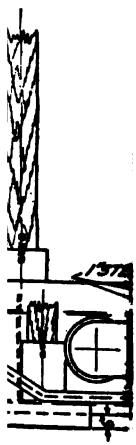
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(3) ROOF SUPPORTS AND PIERS.

Roof supports shall be wooden posts 8 by 8 inches, as designated on Plates XIII and XIV. The posts shall rest on concrete piers, as shown on the drawings, and shall be centered by one dowel pin, $\frac{1}{2}$ inch by 6 inches, in each pier. For the convenience of the Contractor piers may be poured in two sections, one pouring bringing the pier flush with the surface of the ground, after which the reinforcement fabric for the bottom may be placed in position, and the second pouring may be made, bringing the top of the pier to the desired grade and having the floor fabric embedded in it at the proper point so that fabric may form a continuous reinforcement for the floor slabs (see paragraph 5, section IV). The tops of the piers on the bottom of the reservoir shall be in the same plane, and the piers shall be kept damp for at least six days after having been poured before posts may be erected on them.

(4) ROOF GIRDERS.

The girders shall be of 4-inch by 14-inch material, and of lengths as shown on Plates XIII and XIV. They shall be fastened to the posts with $\frac{1}{2}$ -inch by 18-inch drift pins. Girders and posts shall be fastened together by knee braces, as shown on the drawings, of 2-inch by 6-inch material. Knee braces shall be bolted to posts and girders by at least two $\frac{1}{2}$ -inch carriage bolts. Corbels, as shown on Plate XIV, shall be placed on tops of posts in rows 2, 3, 4, and 5, and shall be drifted through with $\frac{1}{2}$ -inch by 24-inch drift pins, as shown.

(5) RAFTERS AND MUD SILLS.

All rafters shall be of 2-inch by 8-inch material, and of lengths as shown on drawings. The outer row of the rafters shall rest on a 2-inch by 12-inch redwood mud sill, as shown on Plate XIII, and shall be notched so as to bear on the full width of the mud sill. A line of 2-inch by 3-inch double bridging shall be placed between the rafters in the middle of each bay, with the exception of the bays of the outer row. In the outer row of bays three lines of such bridging shall be placed as directed by the Company. Bridging shall be nailed in position before sheathing is put on, and two 10d nails shall be used in the end of each bridging. The rafters shall be toe-nailed to the girders with 10d nails, using at least 3 at each rafter. Rafters that lap shall be thoroughly spiked together with 20d spikes, and where joints abut shall be spliced with 2-inch by 8-inch material and thoroughly spiked. The space between rafters at the embankments shall be completely closed by 2-inch by 8-inch material.

(6) SHEATHING.

The roof shall be sheathed with 1-inch by 8-inch sheathing surfaced on one side, furnished in assorted lengths from 8 feet to 16 feet. Sheathing to be No. 1 material, free from knots, pitch pockets or shakes. The roof shall be laid, as shown on Plate XIV, to break joints with roof paper.

Before the sheathing is laid the top of rafters shall be trimmed off with adzes where necessary to insure that sheathing will rest securely on each rafter and will lie smooth and flat. A space of approximately $\frac{1}{4}$ inch shall be left between each board to allow for swelling and shrinkage. Each board shall be nailed with at least three 8d nails on each rafter. All the roof supports and the sheathing, with the exception of a ring approximately 10 feet in width around the circumference of reservoir, necessary for handling the concrete materials, shall all be placed in position before any of the concrete material other than the post footings shall be poured.

(7) VENTILATOR, LADDERS, WALK, HATCH, ETC.

One ventilator, as shown on Plate XIV, shall be constructed on the roof at the center of the reservoir. Contractor shall also place a stairway from the bottom of the reservoir to the top on the inside as designated on Plate XII, and one stairway from the outside toe of the slope to the top of the embankment, to be located in the field by the Company. A wooden walk way shall be constructed by the Contractor from the edge of the roof to the center, having branches to the winch boxes. This wooden walk way shall be composed of 2-inch by 4-inch stringers, and 2-inch by 8-inch planks 2 feet wide, laid with intervening spaces of about $\frac{1}{4}$ inch. The roof of the tank is also

to be equipped with a properly constructed hatch placed at the head of the stairway as shown on Plate XII.

(8) NAILS.

The contractor shall furnish all nails required in the work, to be of sizes as designated in the specifications and drawings.

(9) ROOFING PAPER.

When the woodwork of the roof has been completed, before any roofing paper shall be placed in position, the sheathing shall be thoroughly swept and cleaned of all sawdust, dirt, gravel, chips, shavings, or other material, particular care being taken that no nails are left lying on, or partly imbedded in, the roof. All loose or projecting knots shall be removed, and knot holes exceeding 1 inch in diameter shall be covered with tin plates securely nailed on. The sheathing shall then be covered and made water-tight by placing one layer of 3-ply roofing paper of a quality to be approved by the Company or its duly authorized representative. This paper shall be laid in accordance with Plate XIV. Laps of paper shall be thoroughly cemented together and then nailed with No. 12 roofing nails, the maximum spacing between nails to be 1½ inches.

After the roofing paper has been laid in place, no unnecessary material of any sort shall be wheeled or carried over it, or stored upon it, and as little walking as possible shall be done upon it.

(10) ASPHALT AND GRAVEL COATING.

When the roofing paper has been placed in position as above specified, a heavy coating of hot asphalt, at least 3½ pounds per square yard, shall be applied, and a coating of hot gravel shall immediately be applied to the asphalt. As much gravel shall be used as is held by the asphalt. After cooling, the excess loose gravel shall be removed. Particular care must be taken that hot gravel is applied to hot asphaltum and that a thorough bond is made between the gravel and the asphalt. If such a bond is not made in any part of the covering, the Contractor shall remove that part, including the roof paper, and shall replace it to the satisfaction of the Company.

(11) ROOFING GRAVEL.

The roofing gravel shall be furnished by the Contractor. It shall be composed of pebbles rejected by a screen having ¾-inch openings and passing through a screen having ⅜-inch openings. Gravel shall at all times be subject to inspection by the Company or its duly authorized representative.

(12) WINCH BOX.

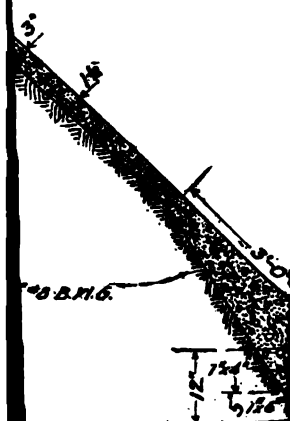
The Contractor shall install on the roof of the completed tank, as directed by the Engineer, two (2) standard iron winch boxes which will be furnished on the ground by the Company.

(13) DRAINS AND GUTTERS.

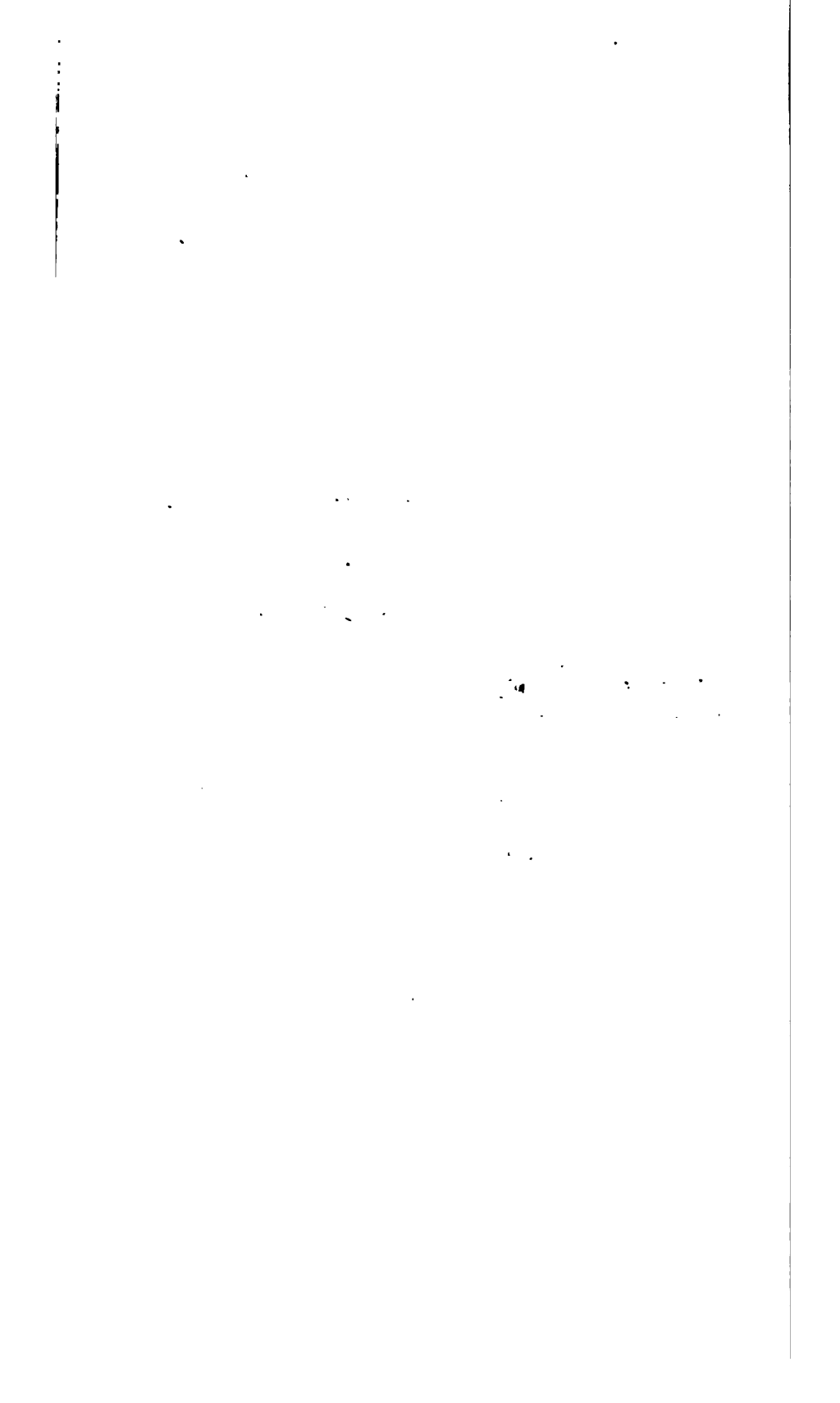
Gutters shall be constructed entirely around the outer edge of the roof, as shown on Plate XIV. The drains from these gutters and downspouts leading from the outer slope of the embankment shall be constructed as shown on Plates XIII and XIV. The galvanized-iron drains shall be furnished by the Contractor and shall be of No. 24 gage iron. The openings leading from the gutters to the downspouts shall be thoroughly and carefully flashed with roofing paper, and all joints shall be covered with hot asphaltic cement so as to make a water-tight job. All drains leading down the slope of the embankment shall be constructed of wood, as per detail drawing and shall extend at least 10 feet beyond the toe of the slope.

(14) CLEANING UP.

Upon completion of the roof, the bottom of the tank and the inside slopes shall be thoroughly cleaned of all small pieces of wood, roofing paper, nails, refuse, etc., which shall be disposed of by the Contractor as directed by the Company.



10,000-BARREL RESERV



SECTION IV.—CONCRETE.

1) NATURE OF THE WORK.

The work to be done by the Contractor under this section consists of constructing upon the sides and bottom of the reservoir, in a thoroughly workmanlike manner, a lining of concrete 3 inches in thickness, reinforced with metallic fabric as herein-after specified, and so carefully poured and compacted as to be as nearly as possible water-tight. Under no condition shall any of the concrete for the sides or floor be poured until the framework and sheathing of the roof, with the exception of the 10-foot strip around the circumference of the reservoir as hereinabove specified, shall have been erected. The piers for roof supports must, of course, be constructed before the section of the roof, as provided in paragraph 3 of section III.

2) MATERIALS, LABOR, TOOLS, ETC.

The Contractor shall furnish all materials, labor, tools, mixers, barrows, carts, mrs, skips, engines, teams, wagons, etc., and any and all other tools and equipment necessary to perform the work as herein described.

(a) CEMENT.

A good grade of Portland cement satisfactory to the Company or its duly authorized representative shall be furnished by the Contractor. Cement may be supplied either in barrels or in sacks, at the option of the Contractor. It shall, however, at all times be subject to the inspection of the Company, the Company reserving the right to reject any and all cement that test shows to be not up to requirements.

(b) SAND.

The Contractor shall furnish the necessary sand for the contract. The sand must be clean and sharp and free from earth, clay, shell, gypsum, mica, or other injurious material.

(c) ROCK.

Crushed rock shall be furnished by the Contractor in sizes as follows:

One-half-inch rock, which shall be retained on a $\frac{1}{2}$ -inch mesh screen, and shall pass through a $\frac{3}{4}$ -inch mesh screen.

One-inch rock, which shall be retained on a $\frac{1}{2}$ -inch mesh screen, and shall pass through a 1-inch mesh screen.

It shall be of a quality approved by the Company, and shall contain no clay or earthy substances or volcanic ash.

(d) REINFORCING MATERIAL.

The reinforcing material used in the bottom and on the sides of the tank shall be welded metal fabric, having a 4-inch-by-4-inch mesh, and composed of No. 6 gage wires both ways. This fabric shall be furnished in rolls 86 inches in width and approximately 240 feet long. The Contractor shall supply, cut, and shape the reinforcing fabric as shown on Plate XV.

(3) WATER FOR CONCRETE.

The water for concrete shall be furnished by the Company and shall be supplied through the water mains surrounding the outer circumference of the reservoir, as provided for in paragraph 7, section II. The Contractor shall furnish and lay all necessary laterals from these water mains to the various positions along the edge of the reservoir where he may have his concrete mixers in operation.

(4) PREPARATION OF THE RESERVOIR FOR CONCRETE.

The final preparation for the pouring of the concrete shall be made by carefully bringing all parts of the bottom and inside of the reservoir to the finished grade. The Contractor shall also at this time excavate for the joint at the toe of the slope and shall remove all material so excavated, together with all material taken off in finishing to exact grades from the inside of the reservoir.

(5) PLACING REINFORCEMENT.

All reinforcement fabric for the bottom of the reservoir shall be laid before roof is put in place, as specified in section III, paragraph 3, and must extend continuously along the bottom and sides of the reservoir through the foundation for the posts. Longitudinal splices shall be lapped at least one mesh and shall be wired together with No. 16 gage annealed iron wire to be furnished by the Contractor. All reinforcement must be properly stretched and made to lie flat. Transverse splices in reinforcement shall lap 24 inches. When the reinforcement is in place and before pouring is begun, care must be taken that all stakes, pins, bolts, etc., used in laying and stretching the fabric are removed. The fabric must be so placed over the entire area that it can readily be pulled into position in the center of concrete slab as the pouring progresses. Arrangement of reinforcing material is shown on Plate XV.

(6) JOINT AT TOE OF SLOPE.

A wooden joint of 1-by-6-inch material shall be placed in the concrete at the toe of the slope inside the reservoir, as shown in detail on Plate XII. The metal fabric shall be laid upon the slopes and the bottom of the reservoir shall pass through the wooden frame in the joint, as shown on the drawing, and shall be so laid as to form a continuous reinforcement, tying sides and bottom together.

(7) PROPORTIONING CONCRETE MATERIAL.

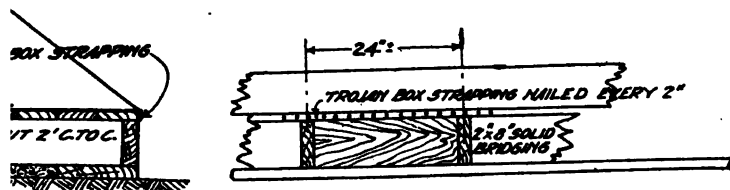
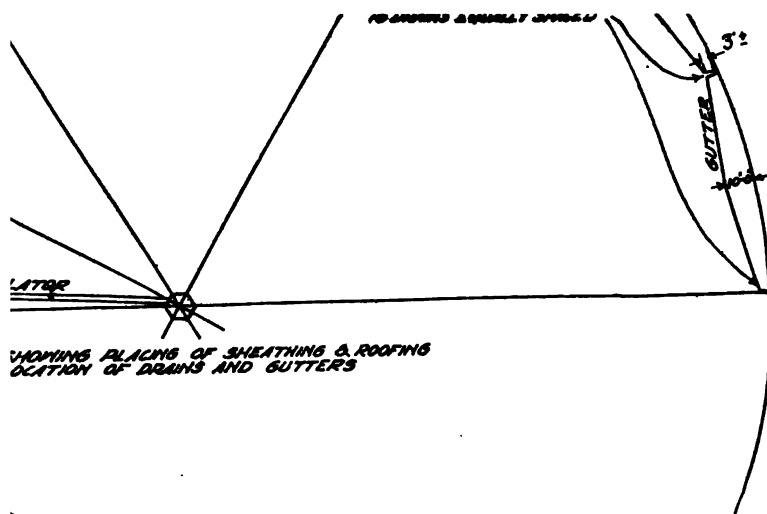
The concrete for the bottom of the reservoir and for the piers supporting the roof shall be composed of one part of Portland cement, two and one-half parts of sand, two parts of $\frac{1}{2}$ -inch crushed rock, and two and one-half parts of 1-inch crushed rock. The concrete for the sides of the reservoir shall be composed of one part of Portland cement, two and one-half parts of sand, and two parts of $\frac{1}{2}$ -inch crushed rock. The proportions of the various materials shall be very carefully measured at all times by the Contractor. The Company, however, reserves the right to change from the proportion as it sees fit, and no such changes shall in any way be construed to affect the contract price.

(8) MIXING AND POURING CONCRETE.

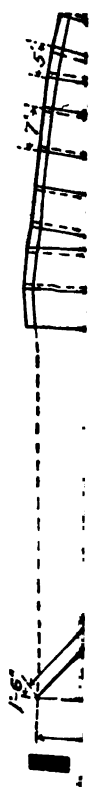
The bottom of the reservoir and the side slopes shall be thoroughly wetted before any concrete material is poured on them. In sprinkling, however, the Contractor shall take care that no reinforcing fabric becomes dirty or muddy; and if any dirt does adhere to the fabric it shall be thoroughly removed before any concrete material is poured on it. The mixing shall be done in batch mixers of a form and type approved by the Company. No more water shall be added to the material than is absolutely necessary to cause the concrete to spread properly over the area to be covered. On both the floor and the side slopes a layer of concrete approximately $1\frac{1}{2}$ inches thick shall first be poured. The reinforcing fabric shall then be drawn up through the concrete and allowed to rest on top of it. As soon as the fabric has been drawn up and before the first layer of concrete shall have obtained its initial set, a second layer shall be immediately poured, bringing the slab up to the required thickness. When the pouring has been completed, the surface of the concrete must be immediately tamped with wooden tampers and then finished smooth and close-grained by troweling. The Contractor must at all times exercise especial care that the position of the wire fabric is as nearly as possible in the center of the slab. He must also furnish men experienced in troweling and floating the surface of the concrete, in order that it may be made as nearly waterproof as possible.

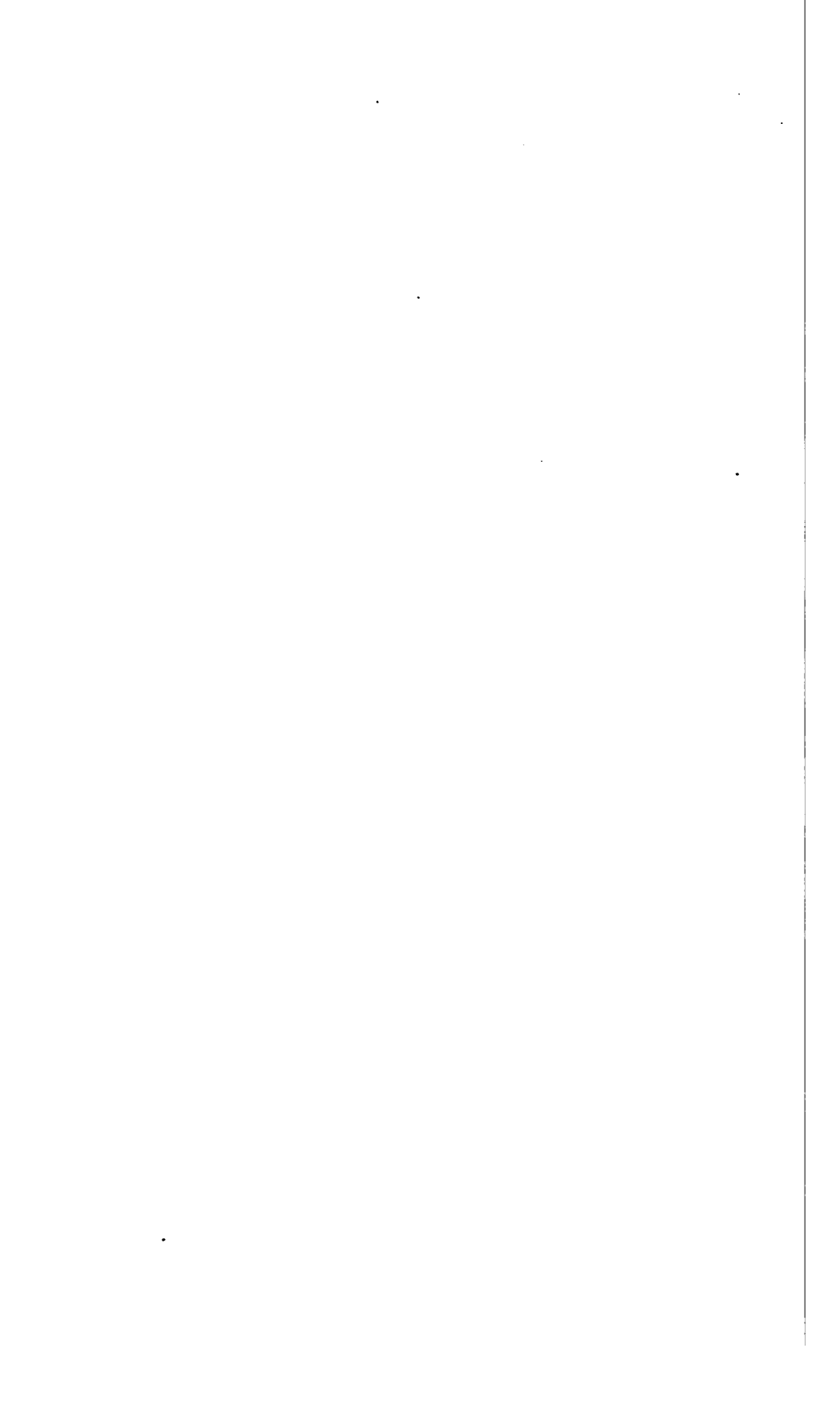
(9) JOINTS IN CONCRETE.

Upon the completion of a day's pouring the edges of concrete slabs shall be roughened and beveled off. When work is resumed, this beveled edge shall be thoroughly cleaned, dampened, and well brushed with neat-cement grout before the next section of concrete is poured.



DETAIL OF ROOFING AT EAVES





(10) SPRINKLING FINISHED WORK.

The Contractor shall keep all of the finished concrete moistened as directed by the Company until the final completion and acceptance of the reservoir by the Company.

(11) GROUTING:

Should small shrinkage cracks appear in any part of the floors or slopes at any time during the concreting, they shall be filled with a cement grout as directed by the Company. After the concrete has been allowed to stand for a period of 20 days, both the floor and the slopes shall be grouted with cement wash of a proportion to be determined by the Company or its duly authorized representative.

GENERAL DETAILS REGARDING RESERVOIR CONSTRUCTION.**CHOICE OF SITE FOR RESERVOIR.**

One of the most important factors upon which the success of a reservoir depends is the judicious choice of the site. The soil should be of homogeneous texture, preferably a sandy clay in which clay predominates. The site should be well drained in order that any surface waters that may percolate through the embankment and get behind the concrete lining may be carried away and not allowed to stand, thereby placing on the sides and bottom a hydrostatic pressure from without that may cause the concrete to crack when the reservoir is empty. It has been suggested that in large reservoirs drain tiles insuring the prompt removal of surface water might well be placed beneath the excavation. No site should be chosen such that the level of adjacent standing water, such as backwater from the arm of a river or bay, will at any time be above the level of the bottom of the completed reservoir. When strata or pockets of sand or other loose material extending into the subgrade are encountered they should invariably be removed for a distance of 3 feet or more below the finished excavation and then refilled to grade with selected earth, put on in thin layers and carefully tamped.

BUILDING EMBANKMENT.

In building up the embankment the "sheep's foot" type of roller tamper (Pl. X, *B*) has proved most effective. A "road machine" or road grader has been successfully used by some contractors for spreading the material, and a heavy harrow or a drag scraper has been used for breaking clods.

Wetting down the loose material is effective in enabling the tampers to build a compact embankment, in making the earth ride satisfactorily in the scrapers, and in keeping down the dust. The usual method of sprinkling is shown in Plate X, *C*.

On completion of the main embankment and the refill on the inner slope it is necessary to trim from the inner slope a foot or more of material to bring it to the true grade and to prepare it for the concrete lining. Grade stakes are set on radial lines at the top and at the inner toe of the slope at distances of approximately 10

feet apart around the circumference of the reservoir. Narrow trenches about 6 inches wide are then dug to grade by the use of mattocks and slope-level boards and 2-inch by 4-inch strips 38 feet long placed flat against the finished surface. The trimming between these slope strips is then done by hand or by especially devised scrapers ^a (Pl. XVI, A and B).

OUTSIDE SLOPE.

The outside slope of the embankment, shown in Plate XVII, A, when finished should be thoroughly sprinkled with oil, which not only prevents the washing of the earth by heavy rains but retards the growth of grass and weeds which when dry become a serious fire menace. It is good practice when the reservoir is in use to oil the slope two or three times a year, or often enough to keep it free from vegetation. A rapid and successful means of oiling is shown in Plate XVII, A. An ordinary 500-gallon tank wagon arranged with a sprinkler spout having an extended arm on one side so that the outer perforations will deliver the oil well toward the toe of the embankment is used. The lower part of the slope is reached from the ground surface.

ROOF.

As previously stated, the roofs of all the large reservoirs so far constructed have been of wood covered with roofing paper. In the opinion of the writer this type of construction is unsatisfactory, as it not only permits the sun's rays to heat the upper surface of the oil but it offers no resistance to the passage of the light hydrocarbon compounds that rise from the body of the oil by reason of such heating.

A reinforced-concrete roof would certainly be a much more permanent type of construction, and a concrete roof made sufficiently strong to uphold a foot or more of earth and tight enough to shed seepage water would be an ideal construction. By this means a low and practically constant temperature could be maintained within the reservoir at all times, and injuries to the concrete lining through expansion and contraction and losses by evaporation would be reduced to a minimum.

Various stages of the roof construction are shown in Plates XVII, B, XVIII, A, and XVIII, B.

CONCRETE LINING.

Too great stress can not be placed on the necessity of thoroughly mixing and tamping the concrete lining, as imperviousness of the lining is quite as important as a proper backing for it. It is abso-

^a Cole, E. D., Concrete-lined oil-storage reservoirs in California; construction method and cost data; Proc. Am. Soc. Civ. Eng., vol. 41, August, 1915, pp. 1347-1350.



A. TRIMMING SLOPE BY HAND.



B. TRIMMING SLOPE WITH SPECIAL SCRAPER.



A. OILING THE COMPLETED EMBANKMENT.



B. INCOMPLETED TANK, ROOF CONSTRUCTION BEGUN AND REINFORCING MATERIAL IN PLACE.



C. EXCAVATING FOR JOINT BETWEEN FLOOR AND SIDE OF CONCRETE LINING.

lutely essential that the finished concrete be dense and have a dense close-grained glazed surface. To obtain this result a mechanical analysis of the aggregates to ascertain their proper proportions in the mix should be taken repeatedly during the progress of the work. The method for determining the proper proportions, described by Fuller and Thompson,^a has been successfully used.

EXPANSION JOINTS.

It will be noticed that the foregoing specifications make no provision for expansion joints in the concrete other than the joint between the floor and the side slopes formed by the small wooden form of L cross section. This is not strictly an expansion joint, as the reinforcement is carried through it from the sides to the bottom. It forms a convenient line to which to pour, and in the more recently constructed reservoirs the form has been removed entirely before the slope concrete has been poured. Excavation for this joint is shown in Plate XVII, *C*.

In the first reservoirs constructed various types of expansion joints were used, but none has proved satisfactory. In fact, more than one company has spent a considerable sum of money in removing entirely the expansion joints from some of their reservoirs, which they found to be leaking badly.

Reed,^b in a discussion before the American Society of Civil Engineers, stated that it is his "belief that expansion joints are not necessary in work of this type, provided there is sufficient reinforcement in the slab and the concrete is carefully cured." The writer is inclined to agree with Reed, especially if the reservoir is comparatively well filled at all times, and when fresh oil is added in any appreciable amount care is taken to see that it is reasonably cold and is delivered into the reservoir well toward the center of the oil already contained and not against the concrete lining. Even when it is necessary to withdraw all the oil from the reservoir there is no reason why a sufficient volume of water can not be placed in it to keep the air at all times cool and moist.

^a Fuller, W. B., and Thompson, S. E., The laws of proportioning concrete: Trans. Am. Soc. Civ. Eng., vol. 59, December, 1907, pp. 67-143.

^b Reed, B. J., Concrete-lined oil-storage reservoirs in California; construction methods and cost of data: Proc. Am. Soc. Civ. Eng., vol. 42, February, 1916, pp. 229-232.

TEMPERATURE VARIATIONS.

Temperature variations throughout the year in a storage reservoir filled with oil are surprisingly small, as is indicated by Table 3 following:

TABLE 3.—*Temperatures without and at various points within K. T. & O. Reservoir No. 2, Bakersfield, Cal.*

[Capacity of reservoir, 500,000 barrels; depth, 21 feet; depth of oil, 19 feet 9 inches (approximately); gravity of oil, 14° B.]

Date.	Atmospheric temperature, °F.	Temperature of oil, °F.			
		2½ feet from bottom.	7½ feet from bottom.	12½ feet from bottom.	17½ feet from bottom.
1914.					
June 1	76	57	57	60	65
July 1	87	62	62	62	69
Aug. 1	88	60	60	66	73
Sept. 1	80	62	61	64	72
Oct. 1	80	62	62	67	72
Nov. 1	62	64	64	65	68
Dec. 1	48	59	59	59	59
1915.					
Jan. 1	37	56	56	56	56
Feb. 1	50	54	53	53	53
Mar. 1	48	54	54	54	54
Apr. 1	58	53	53	54	55
May 1	56	55	55	55	59
June 1	70	56	56	56	62
July 1	80	57	59	63	62

Extreme variation in atmospheric temperature, °F..... 50

Extreme variation in temperature of oil 2½ feet from bottom, °F. 9

Extreme variation in temperature of oil 7½ feet from bottom, °F. 11

Extreme variation in temperature of oil 12½ feet from bottom, °F. 14

Extreme variation in temperature of oil 17½ feet from bottom, °F. 20

Temperatures taken at 7 a. m.

The atmospheric temperature was taken at 7 o'clock in the morning, so that the extreme variation in atmospheric temperature shown is not the maximum variation. The highest temperature for the period was 106° F. at 3 p. m. on August 1, 1914, and the lowest 32° F. at 4 a. m., January 1, 1915, so that the actual extreme variation in atmospheric temperature was 74°. It is safe to say, however, that the highest and lowest temperatures recorded for the oil near the bottom of the reservoir represent very nearly the actual conditions for the period.

EFFECT OF OIL ON CONCRETE.

Where proper care has been taken to make a dense compact lining in which all coarse material is well embedded below the surface there seems no doubt that an oil-tight structure can be made. The writer has repeatedly seen samples of concrete lining taken from the bottoms of reservoirs 20 feet in depth which have been in use for a period of



A. LIFTING ROOF SUPPORTS INTO POSITION.



B. SHEATHED ROOF READY FOR ROOFING PAPER.

five years. These have shown practically no penetration whatever. Mr. H. B. Truett, chief engineer of the fuel-oil department of the Southern Pacific Railroad, states that he examined a 750,000-barrel reservoir belonging to the company after it had been in continuous use for several years, and found the concrete lining in practically perfect condition. A scratch with a penknife removed the oil from the surface of the lining and showed the concrete white and unattacked in almost every place examined, and he found numerous places where the oil could be wiped from the surface of the lining, leaving it scarcely discolored and as smooth and firm as the day it was put in.

W. E. Perdew, chemist for the same company, has made a study of the action of oil on oil-proofing materials and on what was determined by mechanical analysis of the aggregates to be a desirable mix of concrete.

The following abstract of a report made by him shows the results of his experiments:

Concrete blocks 6 by 12 by 3 inches were made from the same materials mixed in the same proportions as were used in reservoirs Nos. 3 and 4, section 7, Kern River. This mix in parts by volume was as follows:

Aggregate.	Proportion.
Cement.....	1.0
Sand.....	2.5
$\frac{3}{4}$ -inch rock.....	1.5
1 $\frac{1}{4}$ -inch rock	3.0

After these bricks had been thoroughly cured, dried, and brushed they were painted with various commercial and noncommercial oil-proofing compounds. They were then submerged, with one smoothly troweled untreated brick, under a 20-foot head of 14° B. oil kept, by means of steam coils, at a temperature of 120° F., and allowed to remain in the oil for 120 days. The following results were obtained with the non-commercial coatings:

Acid sludge—brick washed free of acid; not effective.

Hydrochloric acid (1:1)—supposed to fill mouths of pores with silica; not effective; makes concrete more permeable.

Sulphuric acid (1:1)—same result as with hydrochloric acid.

Glue and chromic acid—concrete brick painted twice with glue and then with potassium dichromate solution; exposed to sunlight two days and then submerged in the oil; failure.

Lead paint—paint made of red and white lead, linseed oil, and turpentine was applied to a concrete brick. Paint tended to blister and peel. Oil gained access to the brick through ruptures in the coating. Better than some coating materials, but not very effective.

Condensation products of phenol and formaldehyde—two bricks were painted with this mixture, one of which was baked before being put into the oil. Coatings on both failed on account of cracking.

Sorel cements—preparation consisted of powdered magnesium oxide mixed with some filler, such as powdered silica and magnesium chloride solution. The powder and the liquid were mixed to the consistency of thick paint and applied with a brush. The coating cracked so badly under the test that it was practically useless.

Perdew goes on to state that the commercial coatings were no more effective than the noncommercial, and that the smoothly troweled untreated brick showed as little penetration as any bricks tested. His report gives the following data regarding "an attempt to decrease the permeability of concrete by precipitating barium sulphate in it as it is being mixed."

Cement: Cowell.

Sand: Unscreened sand, same as was used in the reservoir on section 7-C, Coalinga.

No. 4: Twelve briquets made from sand and cement, 3:1, using 13.75 per cent water. No additions.

No. 5: Eight briquets made from sand and cement, 3:1, using 13.75 per cent water. 0.1 per cent by weight of powdered barium chloride was mixed with the dry cement and sand, and sufficient potassium sulphate dissolved in the mixing water to precipitate all of the barium as insoluble barium sulphate.

No. 6: Eight briquets made with sand and cement, 3:1, using 13.75 per cent water. 0.2 per cent by weight of powdered barium chloride was mixed with dry sand and cement before wetting. Sufficient sulphate ions added to mixing water to precipitate all of barium as sulphate.

No. 7: Eight briquets made from sand and cement, 3:1, using 13.75 per cent water. 0.5 per cent by weight of barium chloride was mixed with the dry sand and cement before mixing with the water, to which had been added enough sulphate ions to precipitate all of the barium as the sulphate.

All briquets were stored under water for 20 days. Then half of each lot was dried in an oven, weighed, and submerged under about a 12-foot head of 14° B. oil, whose temperature was about 120° F. The other half of each lot was placed in a cool, damp place. At the end of 3 days more, all briquets were broken, with the following results:

Data regarding concrete briquets tested for permeability.

[Total age of all briquets, 50 days.]

Lot No.	Average tensile strength of briquets in moist air.	Average tensile strength of briquets in hot oil.	Per cent by weight of oil absorbed.
	<i>Pounds per square inch.</i>	<i>Pounds per square inch.</i>	
4.....	265	325	6.3
5.....	300	280	7.2
6.....	285	280	7.2
7.....	280	280	6.6

It is evident that this method is useless for decreasing the permeability of concrete. In fact, it would seem that it increases it. The tensile strength of the different lots was not overly definite, owing probably to the short duration of the test. It would seem that the precipitation of the barium sulphate, or the resultant formation of potassium chloride, accelerates the hardening of the concrete, based upon the tensile strength of the briquets stored in the moist air. Also, the tensile strength seems to decrease with the increase of barium chloride and potassium sulphate. That the highest tensile strength recorded was that of the No. 4 briquets, which had been under hot oil, bears out these conclusions, and is accounted for by assuming that the heat hastened the hardening of the concrete, while the oil had not had time enough to disintegrate it appreciably.

Practically the same treatment was given to briquets made from Coalinga sand, finely powdered diatomaceous earth from Casamalia, Cal., and cement in the propor-

tion 15:5:2, and it was found that the tensile strength of the briquets was decreased and the permeability was increased.

The same test was also applied to briquets made with Coalinga sand, 1:3, to which was added in the mixing water 0.0041 per cent iron in slightly acid solution, with practically the same results regarding tensile strength and permeability.

The following tests were made on briquets placed in 14° B. oil (about 10-foot head), 120° F., for a period of 22 days. Tests for tensile strength were taken before the briquets were immersed in the oil.

Sodium silicate solution—causes decrease in tensile strength with increase of amount added; and causes increase in permeability with increase of amount added.

Crude oil (4 per cent to 9 per cent additions)—decreases tensile strength and increases permeability.

In conclusion Perdew states:

The fact is again emphasized that dense concrete is the most impervious to oils of 13° to 18° B., even when the oil is heated to a temperature of 120° F. The penetration of oil through the sides of the bricks that were troweled (the tops of the bricks in the molds), and hence the densest, was the least in every case.

COST OF RESERVOIRS.

The cost of a reservoir such as covered by the foregoing specifications would be 10 to 13 cents per barrel of capacity, dependent on the situation and other governing conditions. On a basis of 11 cents, the cost would be distributed approximately as follows:

Cost of earthwork, cents.....	3.5
Cost of roof, cents.....	3.0
Cost of concrete lining, cents.....	4.5
Total, cents.....	11.0

An unlined earthen reservoir with the same type of roof construction would cost 7 to 9½ cents a barrel, and it is estimated that a concrete-lined reservoir with a concrete roof on concrete roof supports and covered with 2 feet of earth would cost about 30 cents a barrel.

The following figures for labor costs cover the construction of two 750,000-barrel reinforced-concrete-lined reservoirs built at Bakersfield, Cal., during the winter and spring of 1913-14:

Labor costs for two concrete oil reservoirs at Bakersfield, Cal.

Earthwork:

Excavating for embankment, per yard.....	\$0.22
Lining inner slopes with selected material, per yard.....	.51
Finishing floor, per square foot.....	.006
Excavating for pier footings, trenches, etc., per yard.....	.70
Trimming slopes, per square foot.....	.012

Roof:

Hauling lumber from cars (¼ mile), per M.....	1.19
Framing lumber for roof, per M.....	1.60
Erecting roof, per M.....	3.80
Sawing sheathing, per M.....	1.35

Roofing:

Laying roofing paper, per square.....	\$0. 12
Hauling roofing gravel from cars ($\frac{1}{2}$ mile), per ton.....	. 25
Placing asphalt and gravel coating, per square.....	. 32

Concrete lining:

Hauling cement from cars ($\frac{1}{2}$ mile), per ton.....	. 54
Hauling sand from creek bed (2 miles), per yard.....	. 86
Hauling rock ($\frac{1}{2}$ mile from cars), per yard.....	. 50
Laying reinforcing metal on slope, per square.....	. 16
Laying reinforcing metal on floor, per square.....	. 08
Pouring concrete piers, per yard.....	^a 4. 63
Pouring concrete floor, per yard.....	^a 2. 51
Pouring concrete slope, per yard.....	^a 3. 46

The figures given are based on the following conditions:

Situation of reservoir, half mile from railroad.

Formation of soil, light sandy clay.

Excavators used, wheel and "fresno" scrapers.

Hours worked a day, 9.

Wage paid laborers, \$2.50 a day.

Wage paid carpenters, \$3.50 a day.

Wage paid concrete laborers, \$2.75 a day.

Wage paid concrete finishers, \$4.50 a day.

Wage paid foremen, \$6 a day.

LIFE OF RESERVOIRS AND STEEL TANKS.

A properly constructed concrete-lined reservoir with concrete roof would last for an indefinite period. The life of the modern steel tank is variously estimated. However, the consensus of opinion among the majority of oil men seems to be that a commercial steel tank such as is in use to-day, if placed on a carefully constructed foundation, and if properly maintained, should last, with the exception of the roof, for 35 to 40 years. The roof, if of wood, covered with roofing paper, would of course require repeated repairs during this period to keep it waterproof and, even though it were constructed with steel, parts of it might need to be replaced three or four times by reason of deterioration from light hydrocarbon gases and sulphur fumes.

Several oil tanks examined by the writer have been in use for 20 to 25 years, and their shells are to all appearances in almost perfect condition. One tank at the Standard Oil Co. refinery, Baltimore, Md., of 30,000-barrel capacity, has been in continual service for more than 35 years. This is a wrought-iron tank made from plates much lighter than those used at the present time for a tank of this capacity. The rivet spacing is also greater than is used in modern practice, which means that the tank was not designed with what is considered to-day an adequate factor of safety. For this reason for several years past

^a Including cost of rock, at \$1.60 per ton. All material except sand and gravel was furnished by the owner at the Southern Pacific R. R. one-half mile distant from the work.

it has been carried only about two-thirds full, and, so the superintendent states, will be cut down and replaced in the near future. The roof has been replaced several times, but the shell so far as can be seen is sound and in good condition.

LOSSES IN STORAGE.

BY SEEPAGE.

Oil losses by seepage from steel tanks and from well-constructed concrete-lined reservoirs can be considered practically nil. From unlined earthen reservoirs, however, seepage losses may be very large. It is difficult to determine just what these losses are for the reason that most of such reservoirs, as previously stated, are used as emergency containers; the oil put into them coming unmeasured from flowing wells.

However, there is a record of a 500,000-barrel unlined reservoir in the Kern River field, Cal., that had to be abandoned on account of the excessive seepage losses. Although the reservoir was in use only a short time, pits dug subsequently in the bottom disclosed that the oil had already penetrated for a depth of more than 20 feet.

Another company in the same field found that its losses from both evaporation and seepage from heavy oil of 14° to 16° B. stored in a 1,000,000-barrel reservoir over a period of six years, when the reservoir was kept approximately full, averaged 0.58 per cent a month, or 6.96 per cent a year. After the reservoir had been lined with concrete the losses were reduced to approximately 0.176 per cent a month, or 2.112 per cent a year. Assuming that the losses by evaporation were the same for the lined as for the unlined reservoir, which was practically the case, as oil was constantly being put into and withdrawn from the reservoir under both conditions, the saving of oil due to the lining was 4.85 per cent per year. In other words, if the reservoir were kept full the yearly saving at the present market price of oil (about \$1.00) would amount to \$48,500 a year, which, placing the cost of lining at 5 cents per barrel, would pay for that lining in a little more than a year's time.

LOSSES BY EVAPORATION.

If we knew definitely the volume of useful hydrocarbon products that "vanish into thin air" from the wells, the transportation systems, the storage farms, and the refineries of the United States in one single year's time we might well be staggered by the size of the figures. To form some realization of their gigantic proportions one needs only to stand in the vicinity of a flowing well newly brought in, and witness the volumes of gases rising like exaggerated heat waves from the well itself, from the flow tanks in which the oil is being collected, from the sump into which it has overflowed, and from

every surface exposed to the atmosphere. The greatest unmeasured loss, of course, takes place in the vicinity of the well. Nevertheless, this loss continues in a large measure, especially as regards light oils, through the gathering systems, the transportation systems, and the receiving tanks at the refineries, and even from oil of low gravity there is a continual stream of the light hydrocarbons escaping from its surface so long as it remains in storage.

The following data from Port Arthur, Tex., give a comparison of the losses in a wooden-roof tank and in an all-steel tank with tight roof, having one central vent pipe leading the escaping gases to a point outside the fire levee.

Comparative losses of oil from a wooden-roof tank and from a steel-roof tank.

Kind of tank.	Date.	Gravity of oil in tank.	Depth of oil in tank.	Loss.		Time in storage.	Percentage of loss.	Percentage of loss in a year.
				Inches.	Barrels.			
	1915.	°B.	Ft. in.			Mo. days.		
Wooden-roof.....	May 12	55.9	28 11	11½	1,759	1 10	3.39	30.5
	June 22	55.2	27 11½					
Steel-roof.....	May 5	54.7	29 3½	2½	363	1 17	.657	5.23
	June 22	54.6	29 1½					

The highest temperature recorded at Port Arthur between May 5 and June 22, 1915, was 100° F.

Another Texas company found, from "a record of the evaporation of naphtha stored in a wooden-roof tank and in a steel-roof tank for a period of six weeks, that the loss from the wooden-roof tank was 3.1 per cent, while the loss from the all-steel tank was 0.42 per cent." On this basis the annual loss from the wooden-roof tank would be 25.20 per cent, and from the all-steel tank 3.40 per cent, or a difference in favor of all-steel construction of 21.80 per cent. With gasoline at, say, 15 cents a gallon at the refinery this would amount to \$75,500 a year in a 55,000-barrel tank, indicating that it is the height of folly to attempt to store gasoline even temporarily in a wooden-roof tank.

The following information covers tank farms in Oklahoma and Kansas, the oil in the tanks having remained practically undistributed after the original filling.

Comparative losses from wooden-roof a tanks in different districts.

Situation of tank.	Date filled.	Volume of oil.	Dates of adding oil.	Volume of oil added.	Total volume of oil put in tank.	Date of latest observation.	Good oil in tank.	Differences.	Sludge.	Loss by shrinkage (evaporation).	Average yearly loss by evaporation.		Grav-ity of oil at time of latest obser-vation.	Grav-ity of oil when tank was filled.
											Period.	Per cent. whole period.		
Ramona, Okla...	July, 1905	35,568.06	Mar., 1905 Jan., 1914	Barrels: 1,734.07 1,455.93	38,758.09	Nov. 1, 1916	Barrels. 34,153.56	4,604.53	1,112.08	3,492.47	July, 1905, to March, 1906... March, 1906, to January, 1914... January, 1914, to November, 1916.	1.94 .71 .30	32.9	31.3
Jenks, Okla.....	Apr., 1907	35,475.75	Mar., 1909 Apr., 1914	1,591.57 1,644.67	38,711.99	do.....	32,885.39	5,846.60	2,233.17	3,613.43	April, 1907, to March, 1909... March, 1909, to April, 1914... April, 1914, to December, 1916.	2.35 .92 .40	36.9	33.4
Cass, Kans.....	Jan., 1905	35,473.02				Sept. 1, 1907	34,024.34	1,448.68	951.46	497.22	January, 1905, to September, 1907.	.48	32.5	32.5
Neodesha, Kans.	Jan., 1904	34,540.05	Feb., 1905 Jan., 1909	924.85 731.39	36,198.29	Sept. 1, 1910	32,761.19	3,437.10	1,386.19	2,050.91	January, 1904, to February, 1905. February, 1905, to January, 1909. January, 1909, to June, 1911.	1.29 .73 .47	31.3	32.7

a Wooden roof of each tank was covered with roofing paper and sheet iron.

b Gravity on Sept. 1, 1907.

c Volume on Sept. 1, 1907.

d Gravity on Sept. 1, 1906.

A report on ten 55,000-barrel tanks, with construction similar to that of the tanks represented in the foregoing table, stored with Cushing oil in the Cushing field, showed the following loss by evaporation:

Gross stock Nov. 1, 1915..... 540,869.11 barrels of 39.7° B. oil
Gross stock Sept. 1, 1916..... 537,530.77 barrels of 39.1° B. oil

Shrinkage..... 3,338.34 barrels

Yearly loss by shrinkage (evaporation), 0.62 per cent.

Two tanks of Healdton oil stored at Fort Worth, Tex., showed the following results:

Shrinkage in one year of Healdton oil in two tanks at Fort Worth, Tex.

Tank No.	Gross stock Sept. 1, 1915.	Gross stock Sept. 1, 1916.	Shrinkage.	Sludge Sept. 1, 1915.	Sludge Sept. 1, 1916.	Increase in sludge.	Yearly loss by shrinkage (evapora- tion).	Yearly settle- ment of sludge.
	<i>Barrels.</i>	<i>Barrels.</i>	<i>Barrels.</i>	<i>Barrels.</i>	<i>Barrels.</i>	<i>Barrels.</i>	<i>Per cent.</i>	<i>Per cent.</i>
136.....	53,175.50	53,044.20	131.30	425.26	739.57	304.31	0.25	0.57
137.....	53,616.00	53,061.13	554.87	366.98	469.79	102.81	.98	.19

• This oil was allowed to stand in earthen sumps for a considerable time before being stored; hence the comparatively small loss by evaporation.

• Gravity of oil, 38.6° B.

• Gravity of oil, 38.1° B.

• Gravity of oil, 38.4° B.

• Gravity of oil, 27.6° B.

A Pittsburgh, Pa., refinery furnishes data regarding steel tanks of 55,000-barrel and 37,000-barrel capacity, having wooden roofs covered with two-ply roofing paper and over that 22-gage sheet iron. They were used for the storage of Pennsylvania crudes, the results being as follows:

Data regarding evaporation losses in two Pittsburgh, Pa., tanks.

55,000-BARREL TANK.

Date of observation.	Gravity of oil.	Oil in tank.	Remarks.
Sept. 3, 1914.....	°B. 44.0	<i>Barrels.</i> 54,036.12	Pennsylvania crude.
Aug. 14, 1916.....	43.5	53,039.33	
Loss.....	.5	997.79	Loss for period, 1.84 per cent, or for a year on same basis, 0.95 per cent.

37,000-BARREL TANK.

June 19, 1914.....	44.5	36,068.51	Pennsylvania crude.
May 1, 1916.....	43.5	33,929.66	
Loss.....	1.0	2,138.85	Loss for period, 5.93 per cent, or for a year on same basis, 3.15 per cent.

Eight new 37,500-barrel steel tanks, having wooden roofs covered with tar paper, over which was sheet iron, were filled with Illinois crude of an average gravity of 33.1° B. The oil was allowed to remain in storage for a period of four years, showing the following results:

Evaporation losses from eight wooden-roof steel tanks filled with Illinois crude.

Period.	Average gravity.	Average loss in gravity.		Average loss in fluid.	
		° B.	Per cent.	Barrels per tank.	Per cent.
	° B.				
When filled.....	33.13				
First year.....	32.22	0.91	2.75	1,007	2.766
Second year.....	31.77	.45	1.39	476	1.306
Third year.....	31.35	.42	1.32	344	.944
Fourth year.....	31.24	.11	.35	215	.588
Total, four years.....		1.89	5.81	2,042	5.606
Average per year.....		.47	1.45	510	1.402

Data from a Middle-West corporation that handles large quantities of crude petroleum and operates extensively in Texas and Oklahoma show that its oil is kept moving so constantly that exact figures on evaporation losses can not be given, but they are believed to be approximately as follows:

Evaporation losses from Texas and Oklahoma crudes.

Grade of oil.	Gravity, ° B.	Annual loss, per cent.
Distillate.....	43 to 50	2.0
North Texas and north Louisiana crudes (light).....	30 to 40	1.5
South Texas and south Louisiana crudes (heavy).....	18 to 25	.7
Oklahoma crude.....	30 to 38	1.0
Oklahoma crude.....	38 to 41	2.5

The following tabulation, compiled by a California company, shows the losses from fresh oils stored for short periods of time while on the way to the refinery or to market. The tanks in which the oil was placed had wooden roofs, some of which were covered with sheet iron. The length of time during which the oil remained quiescent—that is, without additions or withdrawals—ranged from 5 to 30 days. If the time was less than 30 days the amount shown in the column headed "Loss for 30 Days" was computed on the basis of the total loss during the quiescent period. The tanks under observation were in various fields and the oils ranged in gravity from 13.4° to 28.5° B. Temperature readings were taken three times a day—at 7 a. m., at noon, and at 5 p. m., the average temperature shown being the average of these readings for the period.

The average loss per month varied from 0.06 per cent for the heavy oils to 2.5 per cent for light oils, and the loss per year, based on the monthly loss, varied from 0.72 to 30.07 per cent. The yearly loss, however, is obviously too high, as the monthly evaporation would not remain a constant quantity throughout the year, but would decrease somewhat from month to month.

TABLE 4.—*Loss of oil by evaporation from 13 tanks in California.*

[Capacity of each tank, 55,000 barrels, except as otherwise noted.]

TANK 1, LOST HILLS FIELD.

Month.	Average temperature of oil in tank.	Barrels stored.	Number of days.	Total loss.	Loss for 30 days.	Average per cent of loss on amount stored per month.	Average gravity at 60° F.	
1913.	° F.			Barrels.	Barrels.		° B.	
August.....		53,300	11	191	520		22.9	Record of no value.
September.....	81	49,200	9	114	380		23.2	Temperature not given.
October.....	76							Record of no value.
November.....	62	53,000	10	230	690		23.8	
December.....								
1914.								
January.....	63	35,000	11	116	312		23.8	Do.
February.....	63						23.9	
1916.								
January.....	58	5,500	19	76	120		24.6	
February.....	62	10,000	16	76	152		25.7	
March.....	68	9,800	20	114	171		25.5	
April.....	76	8,400	10	76	228		25.4	
May.....	82	48,000	10	116	345		25.7	
June.....								Do.
July.....	92	46,000	15	191	382		26.6	
August.....	94	34,300	11	76	207		27.5	
September.....	90	33,900	17	305	537		27.8	Do.
October.....								
November.....	66	49,200	9	76	253		26.6	
Total.....		445,600		1,765	4,207	0.943		Loss per year, 11.32 per cent.

TANK 2, LOST HILLS FIELD.

1913.								
August.....	96	52,200	22	173	326		23.0	
September.....		50,000	13	107	436		23.3	
October.....	83	62,800	13	754	174		23.4	
November.....	77	54,100	13	340	784		22.7	Looks excessive.

December.....	62	52,000	10	76	228	23.4	Record of no value.
1914.							
January.....							
February.....	61	53,000	10	75½	225	23.1	
1915.							
January.....	56	47,000	9	38	127	24.4	
February.....	60	5,500	24	115	144	24.8	
March.....	66	27,600	18	536	801	24.3	
April.....	76	6,600	30	464	454	23.3	
May.....	80	9,500	11	77	210	25.1	
June.....	88	9,400	14	153	328	25.5	
July.....	92	7,700	12	191	477	26.6	
August.....							Do.
September.....	90	37,500	7	115	492	27.9	
October.....							Do.
November.....	70	51,300	9	153	510		
Total.....		476,300		2,763	5,760	1,209	Loss per year, 14.51 per cent.

TANK 3, LOST HILLS FIELD.

1913.							
August.....	96½	53,500	11	170	463	24.1	Record of no value.
September.....							
October.....	78	49,800	7	57	240	23.4	
November.....	74	54,000	7	228	977	23.3	Do.
December.....							
1914.							
January.....	64	53,000	7	151	648	23.4	
February.....		54,000	16	226	420	24.8	
1915.							
January.....							Do.
February.....	64	10,500	9	76	263	25.7	
March.....							Do.
April.....	76	49,600	13	76	176	25.6	
May.....							Do.
June.....							Do.
July.....	92	13,100	17	191	337	25.9	
August.....	94	9,600	7	38	163	27.1	
September.....	90	10,700	8	115	432	27.2	
October.....							
November.....	76	52,600	13	75	173	27.4	
Total.....		410,400		1,403	4,282	1,043	Loss per year, 12.53 per cent.

TABLE 4.—Loss of oil by evaporation from 13 tanks in California—Continued.

TANK 4, LOST HILLS FIELD.

Month.	Average temperature of oil in tank.	Barrels stored.	Number of days.	Total loss.	Loss for 30 days.	Average per cent of loss on amount stored per month.	Average gravity at 60° F.	
1913.	° F.			Barrels.	Barrels.		° B.	Record of no value.
August.		64,200	5	94	570		22.8	Do.
September.		11,500	21	286	409		22.4	Do.
October.	79							Do.
November.		64	10	226	678		23.5	Do.
December.	64	64,000						Do.
1914.		64	13	151	378		22.4	Do.
January.	64	53,000	13	114	270		22.2	Do.
February.	63	64,000						Do.
1916.		56	11	38	104		23.4	Do.
January.		12,900						Do.
February.		9,300	11	76	207		25.0	Do.
March.	66	49,900	8	76	265		25.3	Do.
April.	78	11,300	8	76	265		25.7	Do.
May.	80	11,300	13	153	353		25.8	Do.
June.	86	11,000	13	153	353		25.8	Do.
July.	94	8,600	13	153	353		26.9	Do.
August.	94	37,300	11	38	104		27.4	Do.
September.	88	11,400	7	76	326		27.6	Do.
October.	76	11,400	7	38	163		27.6	Do.
November.	66	50,100	8	76	265		27.8	Do.
Total.		439,900		1,672	4,770	1.162		Loss per year, 13.94 per cent.

TANK 6, LOST HILLS FIELD.

Month.	Average temperature of oil in tank.	Barrels stored.	Number of days.	Total loss.	Loss for 30 days.	Average per cent of loss on amount stored per month.	Average gravity at 60° F.	
1913.	° F.			Barrels.	Barrels.		° B.	Record of no value.
August.		51,500	16	244	458		22.4	Do.
September.	96	52,000	7	160	643		22.2	Do.
October.		33,700	19	249	718		22.8	Do.
November.	82	53,000	21	249	718		22.8	Do.
December.	76	53,000	31	183	180		23.0	Do.

1914.					
January.....	60	50,000	11	312	23.0
February.....	63	53,000	10	210	22.7
1915.					
January.....	64	4,200	38	38	23.1
February.....	58	4,200	30	76	22.0
March.....	64	4,100	30	115	22.8
April.....	76	4,000	38	38	22.5
May.....	80	3,900	30	76	22.3
June.....	86	3,900	30	153	21.9
July.....	90	3,700	30	115	21.2
August.....	94	2,600	30	76	19.5
September....	88	3,500	30	115	19.0
October.....	74	3,400	30	76	18.7
November....	60	3,300	30	76	18.1
Total.....		385,500	2,527	4,008	1.039
Total, tanks 1 to 5.....		2,167,700	10,120	23,112	1.066
					Loss per year, 12.47 per cent.
					Total average loss per year, 12.79 per cent.

TANK 6, MIDWAY FIELD.

1914.							
January.....	60	38,800	8	57	214	27.2	
May.....	81	49,200	9	253	777	26.3	
Total.....		88,000		290	991	1.126	Loss per year, 13.51 per cent.

TANK 7, MIDWAY FIELD.

1914.							
January.....	58	53,000	10	295	885	28.1	
February.....	62	50,100	12	57	143	28.5	
Total.....		103,100		352	1,028		
					.997		

Loss per year, 11.96 per cent.

TABLE 4.—Loss of oil by evaporation from 13 tanks in California—Continued.

TANK 8, MIDWAY FIELD.

Month.	Average temperature of oil in tank.	Barrels stored.	Number of days.	Total loss.	Loss for 30 days.	Average per cent of loss on amount stored per month.	Average gravity at 60° F.
1913.					Barrels.		* B.
July.....		35,300	18	931	1,551		23.9
August.....		34,000	19	400	631		27.4
1914.							
February.....	62	35,000	12	63	145		28.5
May.....	70	33,400	7	265	1,135		26.0
Total.....		137,700		1,659	3,463	2.506	
							Loss per year, 30.07 per cent.

Looks excessive.

TANK 9, MIDWAY FIELD.

1913.							
August.....		35,100	21	425	609		25.8
October.....	74	34,700	8	120	450		28.4
1914.							
April.....	71	36,100	16	413	775		26.7
May.....	76	32,200	15	269	478		26.1
Total.....		138,100		1,198	2,312	1.673	
							Loss per year, 20.08 per cent.

TANK 10, MIDWAY FIELD.

1913.							
October.....	82	35,300	9	289	980		27.5
1914.							
May.....	89	33,700	7	114	489		25.9
Total.....		69,000		403	1,440	2.100	
Total, tanks 6 to 10.....		835,900		3,992	9,243	1.725	
							Loss per year, 25.20 per cent.
							Total average loss per year, 20.70 per cent.

TANK 11, AT PUMPING STATION NEAR REFINERY.

1913.	(^b)	29,000	84	427	152	0.534	24.0	Loss per year, 0.59 per cent.
August to November.....								

TANK 12,^a KERN RIVER FIELD.

1912.		24,000	31	46			13.4	
October.....		24,000	31	46				
November.....		26,000	30	9				
December.....		26,000	31	8				
1913.								
January.....		26,000	31	13				
February.....		26,000	29	11				
Average per month.....		26,000		89	15	.06		Loss per year, 0.6 per cent.

TANK 13,^a KERN RIVER FIELD.

1912.		26,000	31	26			13.4	
October.....		26,000	31	26				
November.....		26,000	30	21				
December.....		26,000	31	27				
1913.								
January.....		26,000	31	9				
Average per month.....		26,000		85	21	.06		Loss per year, 0.72 per cent.

^a Capacity of tank, 37,000 barrels:^b Average temperature of oil in tank in August, 94°; in November, 70°.

A California company reports that 18° B. asphaltum-base oil placed in storage in a concrete-lined reservoir with wooden roof showed a loss of 5.52 per cent after a year's time, and that during this period the gravity of the oil was reduced to 15.5° B. So far as is known all of this loss was by evaporation. The same company goes on to say that "Fuel oil of more than 14° B. should not be stored in reservoirs. The evaporation and leakage increase in proportion to the rise in gravity. Our experience has been with concrete-lined reservoirs in the Kern field (California) with gravity of oil ranging about 14° B.; out of eight lined reservoirs our smallest average loss in one reservoir was 0.35 per cent a year; our greatest loss in one of these reservoirs for the same period was 1.92 per cent, and the average loss in all the reservoirs for the year was 1.056 per cent."

These reservoirs have wooden roofs covered with roofing paper, and similar in construction to the roof described in the specifications.

Actual figures taken by the writer from the oil-storage reports of another California company show the following yearly losses from three concrete-lined reservoirs.

Evaporation losses from three concrete-lined tanks in California.

Quiescent period.	Capacity of reservoir.	Depth of oil (approximate).	Gravity of oil.	Per cent loss for period.	Per cent loss, same basis, per year.
	<i>Barrels.</i>	<i>Ft. in.</i>	<i>°B.</i>		
Sept. 1, 1915, to Aug. 1, 1916.....	750,000	21 9	14	0.542	2.59
Do.....	750,000	21 10	14	.382	.9
Do.....	500,000	19 10	14	.630	.9

These reservoirs have wooden roofs, and all of the losses indicated so far as is known were from evaporation.

Table 5 following, comprising a compilation of foregoing data, would indicate the average losses by evaporation from various gravities of distillates and crudes. However, the figures can not be taken as a basis on which to estimate closely future losses, for the reason that although most of them seem fairly reliable, it must be remembered that petroleum is an extremely complex mixture of hydrocarbon combinations having various gravities and various boiling points. A crude oil, for instance, with a gravity of 40° B. may be composed of some of the heavier hydrocarbons and some of the very light hydrocarbons, whereas another oil, with the same gravity, 40° B., may be composed of hydrocarbon compounds varying little from that gravity. The first oil would lose rapidly by evaporation when placed in storage, whereas the latter, although originally of the same gravity, would evaporate much more slowly.

The summary does serve to show that there is a considerable loss from all grades of oil when placed in storage, and that the evaporation from some grades of fresh oil is alarmingly large. Although the figures are admittedly incomplete, as the majority of operators have no reliable information as to exact losses from evaporation, it is hoped that publication of the figures may assist to stimulate producers and refiners generally to a more systematic effort toward determining the real magnitude of such losses. When this is done the writer believes the inevitable result will be much more definite and systematic efforts toward overcoming these losses than are at present exercised.

TABLE 5.—Summary of losses of oil in storage by evaporation.

Gravity of oil.	Type of container.	Average monthly loss.	Average yearly loss.	Remarks.
		Per cent.	Per cent.	
60 to 55 distillate.....	Wooden-roof tank, paper top.....		27.8	Plain top; no water seal.
Do.....	All-steel tank.....		4.36	Water topped and lagged.
Do.....	do.....		3.76	
50 to 43 distillate.....	do.....		2.00	
44.5 to 44.....	Wooden-roof tank, sheet-iron top.....		2.02	Pennsylvania crude.
41 to 38 crude.....	All-steel tank.....		2.50	Oklahoma crudes.
38 to 30 crude.....	do.....		1.09	Texas and Louisiana crudes.
Do.....	Wooden-roof tank, sheet-iron top.....		1.61	Kansas and Oklahoma crudes.
33 crude.....	do.....		1.40	Illinois crude.
26 to 18 crude.....	do.....		.70	Texas and Louisiana crudes.
28 to 22 crude.....	do.....			Fresh California crudes.
18 crude.....	Wooden-roof tank, paper top.....	1.30	5.52	Do.
16 to 14 crude.....	Unlined reservoir, wooden roof.....		6.96	Do.
14 crude.....	Concrete-lined reservoir, wooden roof.....		.95	Do.
13.4 crude.....	Wooden-roof tank, paper top.....		.66	Do.

DIFFICULTIES IN OBTAINING RELIABLE DATA REGARDING EVAPORATION LOSSES.

In dealing with large bodies of oil as contained in reservoirs of 750,000-barrel to 1,000,000-barrel capacity the "personal equation" of the gager may make a considerable discrepancy in results. There is also the likelihood of applying an incorrect coefficient of expansion^a in making the necessary volumetric corrections occasioned by reason of the oil having a temperature above or below the normal. For exact work the coefficient of expansion for each oil in question should be determined. Two oils of similar physical characteristics seldom have the same coefficient of expansion. For instance, it is the custom of many companies in handling low-gravity oil to subtract from the volume, if the indicated temperature is above 60° F., and to add to it if below 60° F., an amount equal to 1 per cent of the indicated volume for every 20 degrees variance from the normal temperature, which is another way of saying that a coefficient of expansion of 0.0005 is applied. Suppose, however, that the actual coefficient of expansion was 0.0004 (or such that the applied correction would be 1 per cent for every 25 degrees variance in temperature), and that the reservoir under consideration is of 1,000,000-barrel capacity having the variance in temperature 10 degrees; the computed figures would then be in error by 1,000 barrels.

DEVICES USED FOR LESSENING EVAPORATION LOSSES.

Why does oil evaporate? The accepted theory regarding the constitution of matter and the nature of heat assumes that all liquids are composed of molecules which are held together by mutual attraction. When a liquid is warmed above -273° C. the molecules are in constant and rapid motion, and, as a result there are spaces between them. As heat is applied to the liquid the molecules move more and more rapidly and strike against each other with greater force, separating further against the force of cohesion. Molecules near the surface of the liquid and vibrating rapidly pass out from it. Some are drawn back again into the liquid by cohesion, but some escape into the air above the surface. If the vessel containing the liquid is open, in time all may escape. However, if the vessel is tightly closed the molecules can not escape from it, and even though it be only partly filled with liquid the air above the surface soon becomes so filled with vibrating molecules that the number leaving the surface of the liquid in a given time is just equal to the number going back into it. The air above the liquid is then saturated and evaporation ceases. If the temperature of the liquid in the closed vessel is increased, vaporization again takes place, but ceases as

^a See Crossfield, A. S., The coefficient of expansion of California crude oils and distillates: West. Eng. vol. 6, December, 1915, pp. 229-238.

soon as the air in the vessel above the liquid becomes saturated for that temperature. However, the pressure in the vessel is increased. If the warm saturated air is cooled some of the vapor condenses and returns to the liquid and the pressure is decreased.

Devices for preventing evaporation losses should be designed with two purposes in view—first, to keep the temperature of the oil in the vessel as low as possible, and, second, to make the container as tight as practicable.

WATER-SEAL TOPS.

Perhaps the most common method of decreasing evaporation losses in an oil-storage tank is by the water-seal top, shown in Plate XIX, A. This form of construction is usually applied to small tanks of 1,000-barrel to 5,000-barrel capacity such as run-down tanks in which naphthas and light refined products are received from the tail houses. The construction of the tank is similar to other steel tanks except that it has a flat roof constructed from riveted plate calked and made water-tight, which is placed about 6 inches below the upper edge of the shell. The space inclosed above the plate is then filled with water that can be kept constantly circulating, thus keeping the tank and its contents cool. The tank shown in Plate XIX, A, is also fitted with a relief vent (in the foreground) which permits the escape of gas through a water seal and allows air to be taken in, as when the tank is being emptied, through a swing check valve open to a pressure from without, but closed to pressure from within the tank.

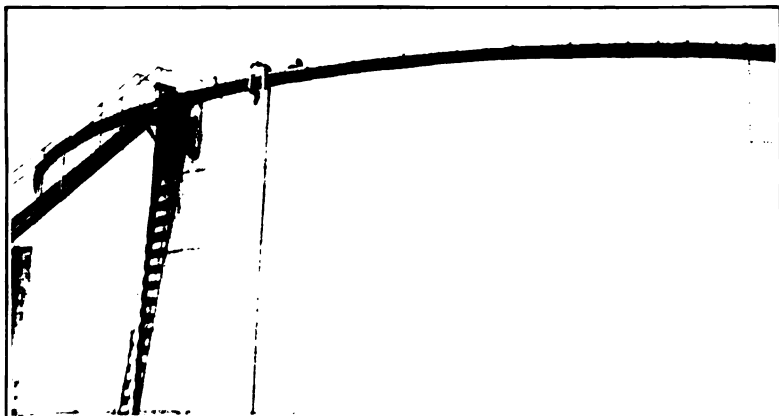
Table 6 following shows the results of tests made with small tanks of 100-barrel capacity, some of which had plain sheet-iron tops and others water tops. Each water-top tank was also protected with a sheathing of 1-inch boards surrounding the sides and the top of the tank and at a distance of about 18 inches from it. The use of these protective devices decreased the losses by evaporation as much as 14 per cent.



A. STORAGE TANK WITH WATER-SEAL TOP.



B. SPRINKLER FOR COOLING TANK.



C. BAFFLE PLATE AROUND EDGE OF ROOF TO DEFLECT WATER FROM SPRINKLER AGAINST SIDE OF TANK. NOTE STEEL STAIRWAY.



TABLE 6.—Results of comparative 30-day tests for losses in plain-top and water-top tanks protected with sheathing.

[Nominal capacity of tanks, 100 barrels; galvanized iron; 9 feet 5½ inches in diameter; 7 feet 10½ inches high 16-gage iron.]

TEST 1.

	Date of test.	Temperature of oil.	Volume in tank.	Corrected gravity.	Losses.		Gravity.
					Barrels.	Per cent.	
Plain-top tank:	1915.	° F.	Barrels.				° B.
Test started.....	Sept. 28	92	92.57	23.4			
Test finished.....	Oct. 27	84	75.02	27.4			
Losses.....					17.55	19	6.0
Water-top tank:							
Test started.....	Sept. 28	86	93.72	33.6			
Test finished.....	Oct. 27	71	90.10	33.3			
Losses.....					3.62	3.96	.3
Saving by use of water top.....					13.93	15.14	5.7

TEST 2.

Plain-top tank:	1915.						
Test started.....	Nov. 17	76	81.43	31.0			
Test finished.....	Dec. 17	64	72.44	26.8			
Losses.....					8.99	11.04	4.2
Water-top tank:							
Test started.....	Nov. 17	71	82.44	30.0			
Test finished.....	Dec. 17	64	82.37	30.0			
Losses.....					.07	.1	.0
Saving by use of water top.....					8.92	10.94	4.2

TEST 3.

Plain-top tank:	1916.						
Test started.....	Aug. 8	90	85.88	29.9			
Test finished.....	Sept. 8	90	73.70	25.7			
Losses.....					12.18	14	4.2
Water-top tank:							
Test started.....	Aug. 8	90	84.66	29.9			
Test finished.....	Sept. 8	78	84.66	29.8			
Losses.....					.0	.0	.10
Saving by use of water top.....					12.18	14	4.1

MEAN TEMPERATURES.

Test 1.—Mean temperatures: 7 a. m., 75.6° F.; 12 m., 101.2° F.; 5 p. m., 80.7° F.; average, 85.8° F. (average temperature not strictly accurate as no midnight temperature was taken).

Test 2.—Mean temperatures: 7 a. m., 47.6° F.; 1 p. m., 62.8° F.; 5 p. m., 60° F.; 12 p. m., 50° F.; average, 55° F.

Test 3.—Mean temperatures: 7 a. m., 90.8° F.; 12 m., 92.5° F.; 5 p. m., 91° F.; 12 p. m., 72.3° F.; average, 86.7° F.

SPRINKLING TANKS WITH WATER.

Plate XIX, B, shows a tank-sprinkling device used by a Pennsylvania company. It is made on the principle of a lawn sprinkler and is placed on the roof at the center of the tank. During hot weather the spraying device is kept in constant operation. The water on

TANK 11, AT PUMPING STATION NEAR REFINERY.

1913.	(1)	29,000	84	437	152	0.324	24.0	Loss per year, 0.59 per cent.
August to November.....								

TANK 12,^a KERN RIVER FIELD.

1912.		35,000	31	45			13.4	
October.....		35,000	31	45			13.4	
November.....		35,000	30	9				
December.....		35,000	31	8				
1913.								
January.....		35,000	31	13				
February.....		35,000	29	11				
Average per month.....		35,000		89	18	.06		Loss per year, 0.6 per cent.

TANK 13,^a KERN RIVER FIELD.

1912.		35,000	31	28			13.4	
October.....		35,000	31	28			13.4	
November.....		35,000	30	21				
December.....		35,000	31	27				
1913.								
January.....		35,000	31	9				
Average per month.....		35,000		85	21	.06		Loss per year, 0.72 per cent.

^a Capacity of tank 37,000 barrels.^b Average temperature of oil in tank in August, 94°; in November, 70°.

3 feet deep is constructed. A swinging pipe can be lowered into the sump when it is desired to drain the entire contents of the tank.

After the forms inside the tank have been removed from the walls and bottom these surfaces are given one to three coats of neat cement. When this has thoroughly dried, several coats of a commercial oil-proofing paint are applied.

One such tank has been in use by an Oklahoma company for the storage of 42° B. oil, since October, 1916. So far the tank has given complete satisfaction and the erection of additional tanks in the near future is proposed.

The estimated cost of concrete construction for various sizes of tank is as follows:

Estimated cost of various sizes of concrete tanks.

Capacity of tank, barrels.	Cost.	Capacity of tank, barrels.	Cost.
250	\$500	5,000	\$3,900
500	775	10,000	7,500
750	1,000	15,000	10,000
1,500	1,750	35,000	15,500
1,800	2,000	55,000	18,000
2,500	2,600		

In connection with the use of underground concrete tanks for the storage of oil it is interesting to note that the El Paso & Southwestern Railroad Co., at various places along its system, has for the past five years been storing fuel oil of 24° to 38° B. in circular concrete tanks about 12 feet in diameter by 6 feet deep. The bottoms of these tanks are 8 inches thick and the sides 6 inches thick, and each tank is covered by a concrete roof. The proportion of the mix is 1:2:4 and the largest rock 1 inch. The aggregate is thoroughly mixed, and care is taken that when it is placed in the forms it is well tamped and all air bubbles thoroughly worked out so as to insure as dense a concrete as possible.

Tanks that have been in use five years have been examined inside and out but no signs of leakage were discovered. At present the company has 12 such tanks, and their adaptability to oil storage has proved so satisfactory that more are in process of construction. No oil or waterproofing compounds are used.

A local company at San Antonio, Tex., has three such tanks, but of much greater capacity (180,000 to 400,000 gallons) in which oils ranging in gravity from 15° to 30° B. have been stored. One of these tanks has been in use 12 years. It was emptied recently and examined inside and out but no signs of leakage were found. The mix, as for the El Paso tanks, is 1:2:4 and the coarsest aggregate 1-inch rock.

At present a fourth tank is being constructed. This will have a diameter of 61 feet 6 inches, a depth of 18 feet, and a capacity of about 400,000 gallons. The bottom will be 17 inches thick and the sides 2 feet at the bottom tapering to 1 foot at the top; all heavily reinforced with steel rods. No oil-proofing compounds are used in any of these tanks.

A local electric company at St. Helena, Cal., built a rectangular reinforced-concrete tank in 1911. This tank is 16 feet long, 14 feet wide, and 5½ feet deep, having a capacity of approximately 10,000 gallons. The bottom is 20 inches thick and the sides 8 inches at the bottom, tapering to 5 inches at the top. Both the sides and the bottom were reinforced with steel rods. At the time the tank was constructed it was coated inside with a solution of sodium silicate. However, after 18 months of service the coating had become practically useless. The tank at this time was examined inside and outside by digging pits, but as no leakage was found no further attempt at oil proofing was made. Since that time the tank has been continually in service for the storage of oil varying in gravity from 15° to 26° B. The roof of this tank is made of wood covered with roofing paper. As a means of lessening evaporation losses and providing a run-off for rain water, as the wooden roof is practically flat, a second corrugated-iron roof of proper pitch has been constructed about 3 feet above the first roof, allowing free circulation of air between the two.

A refining company at Fort Smith, Ark., has two small rectangular reinforced-concrete tanks, one 10 by 14 by 6 feet, and the other 8 by 10 by 5 feet, which have been successfully used for the storage of oils up to 40° B. for the past seven years. There is nothing unusual in the construction of these tanks except that they were made of carefully selected aggregate well mixed and thoroughly tamped. No oil-proofing materials were used.

A large oil company at Tulsa, Okla., has recently completed a concrete reservoir about 200 feet square by 24 feet deep. The walls of the bottom 12 feet have a slope of about 1 : 1, and are supported by the natural earth backing, whereas the walls for the remaining 12 feet are vertical and backed by the excavated earth. The reservoir has a concrete roof, built on much the same principle as the floor of a modern class A building. The intention is to keep the roof of this reservoir covered by 2 feet or more of water and to allow the water to percolate down through the earth along the outside of the concrete.

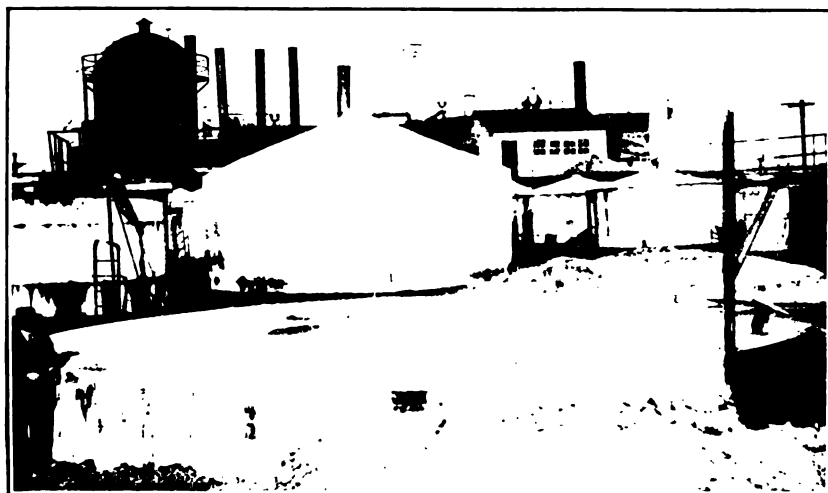
The table following shows a list of companies that are using small concrete containers for the storage of oil, also the kind of oil stored, and whether the container has proved satisfactory.



A. EXCAVATING FOR A CONCRETE-LINED TANK.



B. FORMS FOR CONCRETE TANK.



C. COMPLETED TANK.

List of users of concrete tanks for oil storage.

Name of company.	Address.	Approximate size of container.	When installed.	Gravity of oil stored.	Remarks.
Santa Fe R. R.	Various places along system.	<i>Feet.</i>			
Standard Oil Co.	Clearmont, N. J.	25 by 40 by 12.	1910.	Heavy.	Partly satisfactory.
Do.	do.		1914.	Naphtha.	Walls cracked, owing to settlement in foundation. Two old steel tanks lined with concrete. 1:2:4 mixture.
Williamson Milling Co.	Clay Center, Kans.	12 by 20 by 10.	1910.	Heavy.	Inside given coat of water glass. Satisfactory.
Southern Nebraska Power Co.	Superior, Nebr.	10 by 30 by 6.	January, 1917.	do.	Two-foot earth fill over top. Satisfactory.
Osark Refining Co.	Fort Smith, Ark.	44 by 18.	do.	38° to 40° B., at 60° F.	Satisfactory.
Do.	do.		1912.	Average, 30° B.	Eight small concrete tanks 150 to 260 barrels each. Satisfactory.
Winters Cotton Co.	Winters, Tex.		1909.	Heavy.	50,000 gallons. Given coating 1:1 cement mortar. Satisfactory.
Texas Light & Power Co.	Paris, Tex.	24 by 12.	May, 1915.	28° to 30° B.	Walls 12 inches thick at bottom; 8 inches thick at top. Satisfactory.
Desert Power & Mill Co.	Millers, Nev.		1907.	Heavy.	145,000 gallons. Satisfactory.
Houston Light & Power Co.	Houston, Tex.	34 by 11.	January, 1912.	21° to 30° B.	Floor 10 inches thick. Wall 12 inches; all reinforced. Two compartments.
Cotton Seed Oil Co.	Oklahoma City, Okla.	25 by 10.	September, 1908.	Heavy.	Roof fell in 1914, but tank was otherwise in good condition and still being used.
Magnolia Compress Co.	Houston, Tex.	25 by 16 by 8.	May, 1912.	23° B.	Satisfactory.
Muskogee Refining Co.	Muskogee, Okla.	32 by 11.	December, 1916.	63° B., gasoline.	Three tanks. Partly satisfactory.
San Antonio Gas & Electric Co.	San Antonio, Tex.	45 by 15.	1906.	14° to 21° B.	Wall 36 inches thick at bottom, tapering to 21 inches at top. Bottom 42 inches. Satisfactory.
Do.	do.	24 by 12.	1912.	24° to 28° B.	Satisfactory.
Do.	do.	46 by 16.	1912.	14° to 21° B.	Do.
Do.	do.		May, 1917.	do.	400,000 gallons. Satisfactory.
Seymore Cotton Oil Co.	Seymore, Tex.		1906.	Heavy.	1,200 gallons. Satisfactory.
Anadarko Cotton Oil Mill.	Anadarko, Okla.	45 by 16 by 12.	September, 1915.	28° to 28° B.	Satisfactory.
Fairbanks, Morse & Co.	Beloit, Wis.	35 by 16 by 10.	1916.	Heavy.	Four tanks. Satisfactory.
Ada Gas & Electric Co.	Ada, Okla.	20 by 8.	September, 1908.	28° B.	Used for oil until July, 1911; since for water. Satisfactory.
Charles City Gas Co.	Charles City, Iowa.		1916.	Heavy.	30,000 gallons. Satisfactory.
Shawnee Gas & Electric Co.	Shawnee, Okla.	22 by 10.	November, 1908.	28° B.	Satisfactory.
Cosden Refining Co.	Tulsa, Okla.	200 by 200 by 25.	1916.	Oklahoma crude.	Partly satisfactory.
Dewey Portland Cement Co.	Dewey, Okla.	23 by 7.	1910.	25° B.	Thoroughly satisfactory.
Webster Oil & Gasoline Co.	Oilton, Okla.	43 by 12.	1916.	42° B.	Satisfactory.
Indianola Refining Co.	Oklmulgee, Okla.		1909.		Four tanks. One (10,000 barrels) still in use for hot crude. Only partly satisfactory. Others failed by settling.

a Diameter.

PAINTING TANKS LIGHT COLORS.

Many oil operators, especially throughout the eastern and the middle western fields, have adopted white or light-colored paints for storage tanks. It is reported that tests have demonstrated that evaporation from tanks painted white averages about 1 to 1½ per cent less than from tanks painted red, and about 2½ per cent less than from tanks painted black. These figures have not been verified, but tests made by the Institute of Industrial Research^a show that dark-colored paints absorb heat to a considerable degree, and paints presenting a highly glossy surface are less absorptive of thermal rays than those presenting a matte surface.

Experiments were conducted wherein small tanks containing benzine were painted in various colors (gloss finish) and then subjected to the rays of a powerful arc light for 15 minutes. At the end of that period the rise in temperature of the benzine was as follows:

Rise in temperature of benzine stored in tanks of various colors.

Color of paint or covering.	Rise in temperature, ° F.
Tin plate (tank plated with tin).....	19.8
Aluminum paint.....	20.5
White paint.....	22.5
Light-cream paint.....	23.0
Light-pink paint.....	23.7
Light-blue paint.....	24.3
Light-gray paint.....	26.3
Light-green paint.....	26.6
Red iron oxide paint.....	29.7
Dark prussian blue paint.....	36.7
Dark chrome green paint.....	39.9
Black paint.....	54.0

Tin plating and aluminum paint, as will be seen, gave the best results. However, neither of these finishes is practicable for outside work, as iron coated with tin corrodes rapidly, and aluminum paint soon loses its gloss and becomes flaky. The rise in temperature of the benzine in the tank painted black was 31.5° F. greater than the rise in the tank painted white, or 140 per cent. Although results such as obtained by these laboratory experiments could not be expected in actual practice, there is undoubtedly a decided advantage to be gained by painting storage tanks white.

^a Gardner, H. A., The heat-reflecting properties of colors applied to oil and gas storage tanks: Circular 44, Paint Mfg. Assn. of U. S., Educational Bureau, Sci. sec., January, 1917, p. 1.

COMPUTING VOLUME OF TANKS AND RESERVOIRS AND PREPARING GAGE TABLES.

METHODS OF OBTAINING FIELD DATA REGARDING TANKS.

The methods used by the various oil companies in obtaining the data necessary for computing the volumes of oil tanks and reservoirs, and for preparing gage tables are practically the same, differing only in minor details. As regards steel tanks there is much discussion as to just how many circumference measurements are necessary for accurate work. Some companies take only two (for a 6-ring tank), the lower one being at the middle of the second ring and the upper as

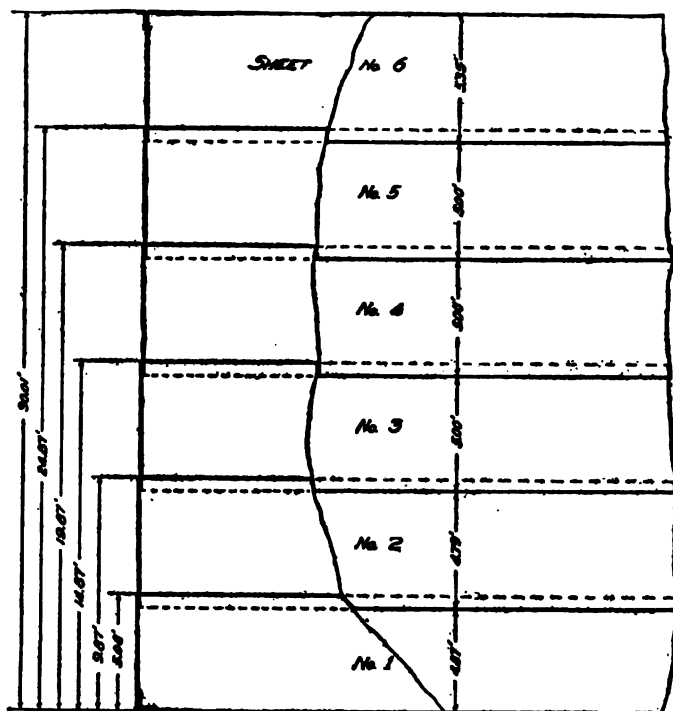


FIGURE 3.—Points between which various measurements for height of sheets apply.

near as practicable to the middle of the top ring. Other companies "strap"^a above each seam, and take one measurement 6 inches below the top of the tank, and one near the bottom just above the bottom row of rivets. However, it has been found that in measuring steel tanks of standard size a surprisingly small variation in results will be obtained if circumference measurements are taken on each ring, or if only the top and bottom rings are measured and the circumference

^a A term in general use among oil operators, originally meaning the girthing of tanks with a measuring tape, but in modern diction includes the whole operation of obtaining the necessary field data for computing the volume of tanks and other fluid receptacles.

of the others computed. On the other hand, tanks on poor foundations causing them to be "out of round" and producing bulging uneven sides may require a large number of measurements to insure accuracy. Although it is difficult to lay down any fixed rules that will apply to all classes of tanks it is believed that the following form of field notes includes sufficient data for reasonably accurate results. A 55,000-barrel tank has been assumed in which the roof supports consist of wooden posts tied together with sway bracing and resting on footing blocks, as this is the most complicated form of construction so far as volumetric computations are concerned. The points between which the various measurements for height of sheets apply are shown in figure 3.

RECORD OF FIELD NOTES SHOWING MEASUREMENTS NECESSARY FOR COMPUTING VOLUMES OF STEEL TANKS.

Location of tank: Party:
Size of tank: Date:

Measurements of tank^a by courses.

Item.	Course No. ^b					
	1	2	3	4	5	6
Circumference, feet.....	360.37	360.17	360.09	^c 360.13	^c 360.17	360.22
Number of sheets, feet.....	26	26	26	26	26	26
Lap of vertical seam, feet.....	.19	.19	.19	.19	.17	.12
Lap of horizontal seam, feet.....	.21	.21	.21	.21	.21	
Horizontal segment, feet.....	2.50	2.00	2.00	^c 1.58	^c 1.17	.67
Thickness of sheet, inches.....	$\frac{1}{16}$	$\frac{1}{8}$	$\frac{1}{16}$	$\frac{1}{16}$	$\frac{1}{8}$	$\frac{1}{16}$
Thickness of sheet, feet.....	.06	.04	.03	.02	.02	.02
Bottom of tank to top of sheet (outside), feet.....	5.08	9.87	14.87	19.87	24.87	^d 30.01
Edge to edge of sheet (inside), feet.....	4.87	4.79	5.00	5.00	5.00	^d 5.25

^a Bottom angle iron, $\frac{1}{2}$ inch by 4 inches by 4 inches; measurements taken approximately in center of courses.

^b Courses numbered from bottom of tank upward.

^c Calculated.

^d Top of top angle iron.

DEADWOOD.

Footing blocks:

Outside circle posts, 22 pieces, 2 inches by 8 inches by 2 feet. Bottom of tank to 2 inches.

Third circle posts, 12 pieces, 2 inches by 8 inches by 2 feet. Bottom of tank to 2 inches.

Third circle posts, 2 pieces, 2 inches by 8 inches by 2 feet. Bottom of tank from 2 inches to 4 inches.

Second circle posts, 10 pieces, 2 inches by 8 inches by 2 feet. Bottom of tank to 2 inches.

Center circle posts, 2 pieces, 2 inches by 8 inches by 12 $\frac{1}{2}$ feet. Bottom of tank to 2 inches.

Center circle posts, 2 pieces, 2 inches by 8 inches by 12 $\frac{1}{2}$ feet. Bottom of tank, from 2 inches to 4 inches.

Bottom ties:

Outside circle posts, 2 pieces, 2 inches by 8 inches by 15 feet 4 inches. From 4 inches to 12 inches.

Bottom ties—Continued.

Third circle posts, 1 piece, 2 inches by 8 inches by 16 feet 8 inches. From 6 inches to 14 inches.

Second circle posts, 2 pieces, 2 inches by 8 inches by 15 feet. From 2 inches to 10 inches.

Swing-pipe rest:

1 piece, 1½ inches by 8 inches by 13 feet. From 12 inches to 20 inches.

2 pieces, 1½ inches by 8 inches by 20 inches. From 2 inches to 20 inches.

2 pieces, 2 inches by 8 inches by 24 inches. From bottom of tank to 24 inches.

2 pieces, 1½ inches by 8 inches by 20 inches. From 23 inches to 24½ inches.

Sway bracing:

Outside circle posts, 48 pieces, 1 inch by 6 inches by 24 feet. From 3 feet to 21 feet 8 inches.

Third circle posts, 30 pieces, 1 inch by 6 inches by 23 feet. From 3 feet to 20 feet 5 inches.

Second circle posts, 20 pieces, 1 inch by 6 inches by 22 feet. From 3 feet to 22 feet.

Center circle posts, 4 pieces, 1 inch by 6 inches by 18 feet 6 inches. From 4 inches to 14 feet.

Center circle posts, 4 pieces, 1 inch by 6 inches by 18 feet. From 4 inches to 13 feet 5 inches.

Center circle posts, 2 pieces, 2 inches by 8 inches by 14 feet 6 inches. From 13 feet 5 inches to 14 feet 1 inch.

Girders:

Outside circle posts, 24 pieces, 5 inches by 12 inches by 15 feet. From 28 feet 6 inches to 29 feet 6 inches.

Knee braces:

Outside circle posts, 12 pieces, 2 inches by 8 inches by 6 feet. From 24 feet 5 inches to 28 feet 6 inches.

Third circle posts, 16 pieces, 2 inches by 8 inches by 6 feet. From 27 feet 4 inches to top (45°).

Second circle posts, 10 pieces, 2 inches by 8 inches by 6 feet. From 28 feet 6 inches to top (45°).

Posts:

Outside circle, 24 pieces, 6 inches by 6 inches. From 4 inches to 28 feet 4 inches.

Third circle, 16 pieces, 6 inches by 6 inches. From 4 inches to top.

Second circle, 10 pieces, 6 inches by 6 inches. From 2 inches to top.

Center circle, 4 pieces, 6 inches by 6 inches. From 4 inches to top.

Rafters: 120, 2 inches by 8 inches.

Distance from bottom of tank to bottom of rafter at center of tank, 36 feet 6 inches.

Top of rafter at shell flush with top of top angle iron.

Manhole, 23½ inches in diameter by 6 inches deep. Bottom, 20 inches above bottom tank.

METHODS OF MEASURING.

The outside measurements should be taken in feet and hundredths, the most convenient form in which to have the data for computations. Measurements of deadwood, by which is meant all material inside the tank that displaces oil, such as roof supports, blocking, steam coils, and water draw-off pipes, are usually more conveniently taken in feet and inches, as the parts mentioned usually consist of standard-sized timbers and pipes.

The circumference measurements are taken as nearly as possible in the center of each course with the exception of courses 4 and 5 (count-

ing from the bottom upward), which can be more accurately calculated by interpolating between the girths of courses 3 and 6 than by actual measurement. The circumference of the top course is taken by reaching down over the eaves of the tank, the tapeman lying face down on the roof.

If possible, inside measurements should be taken before any oil is put into the tank. Outside measurements should be taken when the tank is full of oil.

After one set of dimensions has been taken it is checked by taking a second set in exactly the same manner. The maximum permissible difference in these two sets of measurement should not be more than 1 in 10,000.

MEANING OF TERMS.

HEIGHT OF TANK.

The height of a tank for strapping purposes is the distance between the top of the bottom plate of the tank and the top of the horizontal leg of the top angle iron.

HORIZONTAL SEGMENT.

By the horizontal segment is meant the part of the tank shell, less the vertical lap, which encroaches on the area inclosed by a circle, the plane of which is parallel to the tank bottom and the diameter of

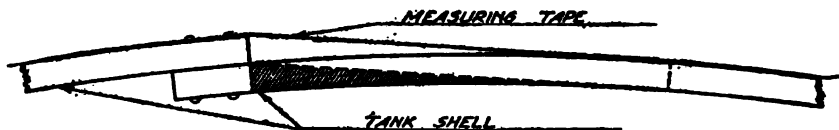


FIGURE 4.—Horizontal segment.

which is equal to the inside diameter of the tank. The hatched part in figure 4 shows the horizontal segment, which is approximately equal in area to the triangle made by the tape and the tank shell. The horizontal segment is not considered at all by many operators and by others is used only when very accurate results are required.

THICKNESS OF SHEETS.

The thicknesses of the sheets in the various rings can usually be most accurately and most easily determined from the specifications of the tank.

DEADWOOD.

The swing pipe and fittings and the water draw-offs in the bottom of the tank are usually omitted for the reason that they are always full of oil or water whenever a gage of the tank is taken, and the metal of which they are composed will at most displace only a small volume of liquid. To guard against the possibility of the swing pipe being empty or only partly filled when a gage of the tank is taken, some companies install on the shell of the tank about 2 feet from the

swing-pipe nozzle a 2-inch flange having its center line on approximately the same horizontal plane as the center line of the nozzle. A 2-inch pipe connection containing a valve is then placed between this flange and the nozzle, so that oil may be admitted from a point near the bottom of the tank into the bottom of the swing pipe whenever desired. Sand lines are also omitted. However, if there are steam coils in the tank, their volume must be deducted.

MANHOLE.

The dimensions of the manhole should be determined, as also its position, as the volume of oil that it contains must be added to the volume of the tank.

CORRUGATED-IRON AND WOODEN TANKS.

As regards corrugated-iron tanks, which are usually of small diameter, the circumference measurements are obtained with the tape lying in the valley of the corrugation. The depth of the corrugation and the distance between corrugations are noted.

In strapping wooden tanks with sloping sides, the thickness of the staves is taken at a number of places about the circumference, and an average thickness determined. The height is obtained by placing a gage rod inside the tank and measuring the vertical height to the top of the staves. The circumference strappings are taken at intervals of 2 feet, measured along the slope of the tank, beginning 6 inches from the bottom.

COMPUTING GAGE TABLES FOR TANKS.

From the circumference of each course the outside diameter of that course is computed, and by subtracting twice the thickness of the metal the inside diameter is determined, giving a value from which the gross area of the course is determined. From this is subtracted one-half the area of the vertical laps and the area of the horizontal segments, giving the net area (plus the deadwood). As the circumference is measured approximately in the center of the sheet, the net area value is considered to be a mean area for the course.

The volume in barrels per course, including the deadwood, is then calculated and the volume in barrels per one-fourth inch per course. Next the volume of the deadwood per course is determined and the volume of the deadwood per one-fourth inch per course. Then a table is prepared showing the quantity in barrels to be added for each one-fourth of an inch proceeding from the bottom of the tank upward. The construction of the gage table then resolves itself into the summation of the proper increments. If it were desired to have the gage table read to one-eighth inch the same method of computation would be pursued. In buying and selling crude oil it is customary to read the gage tape to the nearest one-eighth of an inch and interpolate in the table to find the correct volume.

Each step in the computations should be carefully checked. One-fourth inch totals must equal totals for courses calculated by subtracting the volume of the total deadwood in the course from the total volume of the course, and totals of courses must agree with a separate computation for the whole tank in which figures for total tank volume and total deadwood are used.

As a final check on the whole work a second man is required to make an independent calculation not quite so complete as the first but one that must check the total contents of each course and the total contents of the tank. Gage tables are made to read to the nearest one-hundredth of a barrel.

RESERVOIR MEASUREMENT AND TABLES.

For the convenience of the gager a small hatch is placed in the roof of the tank, usually near the head of the stairway. The gage tape is suspended through the hatch into the open tank. However, as regards reservoirs, it is customary to set what is known as a gage plate at some convenient place as close as practicable to the lowest point in the bottom of the reservoir and to take all measurements for depth of oil from this plate. It comprises a steel plate about $\frac{1}{4}$ inch thick and 3 feet square, which is set with foundation bolts and carefully leveled and grouted. This work is done as the part of the floor in that vicinity is being constructed so that it becomes an integral part of the lining. Some companies use a smaller gage plate and place on it a vertical pipe 4 to 10 inches in diameter of sufficient length to extend out through the roof of the reservoir. The walls of the pipe are perforated or slotted, and the top is fitted with a cap. Measurements for depth of oil are then taken through the pipe.

TESTING GAGE PLATE.

One of the first duties of the field or construction engineer in procuring the data necessary for the preparation of a gage table for the reservoir is to examine carefully the gage plate and determine whether all the foundation bolts are tight, and whether the plate has been properly grouted and has not sprung at any point, but is absolutely level over its entire surface. The following information should then be procured:

NECESSARY MEASUREMENTS IN TESTING GAGE PLATE.

1. The diameter at the toe of the inner slope.^a
2. The diameter at the top of the inner slope.^a
3. The elevation of the gage plate.
4. The elevation of a sufficient number of points on the bottom so that an accurate contour map may be plotted, the contour interval of which should be not more than one-fourth of an inch.

^a This measurement should be taken in as many places as possible and carefully checked.

5. The dimensions of all footings, posts, braces, and other deadwood as outlined for tank strappings.^a

4. The location and the size of the sump containing the swing pipes and water draw-off pipes.

COMPUTING GAGE TABLE.

From the elevation of the points on the bottom of the reservoir a contour map of one-fourth inch interval is plotted and from this the gross content up to the gage plate is computed. To this is added the volume of the sump and from the sum is subtracted the volume of the deadwood up to the plane of the gage plate. The net amount thus obtained gives the content of the reservoir up to 0 feet 0 inches on the gage table.

If the gage table is to read to one-eighth of an inch, as is usually the case, the content is computed in a similar manner proceeding by eighths of an inch, up to the point where a horizontal plane will cut the side slopes of the reservoir. From this point to the top the volume is computed at one-eighth inch intervals, the reservoir being considered as a frustrum of a cone, deadwood being deducted as in the tank-table computations.

Gage tables are made to read to the nearest one-hundredth of a barrel. Volumes for fractions of an inch less than one-eighth are obtained by interpolation.

DEPRECIATION OF OIL IN STORAGE.

Table 7 shows comparative results of distillations of various fresh crude oils and oils from the same fields after being in storage for a period of years. The table indicates that practically every oil showed a considerable loss in the total proportion distilled from the stored liquid and that this loss reached its maximum at an end point of about 150° C.

Table 8 shows comparative results of tests on fresh and stored heavy California crude. The ultimate test showed that this low-gravity oil although having remained in storage for a period of six years seemingly had not lost in fuel value.

Although the samples of fresh and stored oil were taken from the same fields and as nearly as possible from the same wells, the fresh-oil samples were not, unfortunately, taken and analyzed at the time the oil was put in storage; on the contrary they were taken from the wells at the end of the storage period. The fact that the composition of oil produced by a given field often changes somewhat from time to time may account for seeming discrepancies in some of the figures.

^a As regards a reservoir, however, on account of the slope to the bottom, the exact location of the deadwood must also be determined.

TABLE 7.—*Results of comparative distillation tests of fresh and of stored crudes.*

[Apparatus, Bureau of Mines Hempel flask.]

CUSHING CRUDE.

[Stored crude in storage 2 years 5 months.]

Temperature. ° C.	Distillate of fresh crude.			Distillate of stored crude.			Fractions, per cent by volume.		Loss in total per cent distilled.	General data.		
	Frac- tions, per cent by dis- vol- ume.	Total per cent dis- tilled.	Grav- ity.	Frac- tions, per cent by dis- vol- ume.	Total per cent dis- tilled.	Grav- ity.	Loss.	Gain.		Item.	Fresh crude.	Stored crude.
Up to 75.....	6.5	6.5	° B. 83.0	0.5	0.5	° B. 64.0	6.0		6.0	Gravity, ° B.....	40.9	23.0.
75 to 100.....	4.8	11.3	65.5	4.5	6.1	60.3	1.2		6.2	Flash point, ° F.....	Below 75	Below 75.
100 to 125.....	7.2	18.5	60.0	6.2	11.3	60.3	1.0		7.2	Fire point, ° F.....	Below 75	Below 75.
125 to 150.....	6.4	24.9	55.0	8.0	19.3	56.5			5.4	Sand.....	Appreciable amount.	None.
150 to 175.....	6.3	31.2	50.3	6.9	26.2	51.0			5.6	Water.....	None.	None.
175 to 200.....	6.0	37.2	47.0	5.2	31.5	47.0	7		5.7	Viscosity at 20° C., ° Engler.....	1.4	1.4.
200 to 225.....	5.5	42.7	44.0	6.8	38.2	43.8			4.5	Cold test at 0° C.....	Liquid.	Liquid.
225 to 250.....	5.7	48.4	41.0	6.2	44.5	40.5			3.9			
250 to 275.....	5.6	54.0	38.5	6.3	50.8	38.8			3.2			
275 to 300.....	6.5	60.5	36.5	6.7	57.5	36.5			3.0			

GLENN CRUDE.

[Stored crude in storage 7 years 3 months.]

Up to 75	75 to 100	100 to 125	125 to 150	150 to 175	175 to 200	200 to 225	225 to 250	250 to 275	275 to 300	2.7	2.7	Gravity, ° F.	Flash point, ° F.	Fire point, ° F.	Sand	Water	Viscosity at 20° C., ° Engler	Cold test at 0° C.	32.9	Below 72	Below 72	None	None	None	2.1	Liquid.	
2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	
79.0	63.3	59.0	54.0	50.0	45.8	42.8	40.0	38.3	36.3	0.0	0.0	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	
1.3	2.4	6.1	6.0	6.0	6.3	6.7	7.0	7.2	7.8	1.3	1.3	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	
55.0	50.2	46.5	43.5	40.5	37.5	34.3	31.5	28.5	26.5	1.9	1.9	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	
1.1	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.1	1.1	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	
2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	
Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid	Liquid

OSAGE CRUDE.

[Stored crude in storage 7 years 8 months.]

Up to 75.	2.3	81.0	0.0	2.2	2.3	Gravim. "B."	33.7	31.2.
Up to 75.	2.3	81.0	0.0	2.2	2.3	Flash point, °F.	Below 75.	Below 75.
75 to 100.	4.7	5.9	1.6	2.1	4.3	Fire point, °F.	Below 75.	Below 75.
100 to 125.	4.3	10.1	4.8	1.0	5.3	Sand.	None.	None.
125 to 150.	4.2	14.3	3.2	0.0	5.0	Water.	None.	None.
150 to 175.	5.0	19.8	4.8	6	5.9	Viscosity at 20° C.	1.9	2.5.
175 to 200.	4.0	23.7	45.0	2.2	6.1	Cold test at 0° C.	Liquid.	Liquid.
200 to 225.	5.4	28.3	5.8	1	6.2			
225 to 250.	6.0	34.7	6.8	0.8	5.4			
250 to 275.	6.5	41.2	7.0	3.2	4.9			
275 to 300.	5.3	47.0	9.0	3.2	1.7			

BARTLESVILLE CRUDE.

[Stored crude in storage 4 years 8 months.]

[illegible]

NOWATA COUNTY, OKLA., CRUDE.

[Stored crude in storage 7 years 4 months.]

[illegible]

TABLE 7.—Results of comparative distillation tests of fresh and of stored crudes—Continued.

[Apparatus, Bureau of Mines Hempel flask.]

"OEMULGEE COUNTY," OKLA., CRUDE.

[Stored crude in storage 7 years 4 months.]

Temperature.	Distillate of fresh crude.			Distillate of stored crude.			Fractions, per cent by volume.		Loss in total per cent distilled.	General data.		
	Fractions, per cent by volume.	Total per cent distilled.	Grav-ity.	Fractions, per cent by volume.	Total per cent distilled.	Grav-ity.	Loss.	Gain.		Item.	Fresh crude.	Stored crude.
° C.			° B.			° B.						
Up to 75.....	1.0	1.0			0.0		1.0		1.0	Gravity "B".....	33.0	32.4.
75 to 100.....	1.5	2.5			1.0		1.5		2.5	Flash point, ° F.....	Below 72.....	120.
100 to 125.....	3.0	5.5	60.0	1.5	1.5		1.5		4.0	Fire point, ° F.....	111.....	165.
125 to 150.....	3.0	8.5	55.0	2.9	4.4	54.8	.1		4.1	Sand.....	None.....	None.
150 to 175.....	3.7	12.2	51.2	3.6	8.0	51.0	.1		4.2	Water.....	None.....	None.
175 to 200.....	3.8	16.0	48.3	3.0	11.0	48.0	.8		5.0	Viscosity at 20° C., ° Engler.....	2.5.....	4.3.
200 to 225.....	4.3	20.3	45.3	4.7	15.7	45.0		0.4	4.6	Cold test at 0° C.....	Liquid.....	Slightly viscous.
225 to 250.....	5.4	25.7	42.5	5.3	21.5	42.0			4.2			
250 to 275.....	6.5	32.2	39.4	6.5	28.0	39.4			4.2			
275 to 300.....	6.5	38.7	38.8	10.5	38.5	38.7		4.0	.2			

TABLE 8.—Results of comparative tests of fresh and of stored California crude.

DISTILLATION TESTS.

Temperature.	Fresh crude.			Crude stored for 6 years.			Fractions, per cent by weight.		Loss in total proportion distilled.
	Fractions, per cent by weight.	Total per cent distilled.	Gravity.	Fractions, per cent by weight.	Total per cent distilled.	Gravity.	Loss.	Gain.	
*C.			*B.			*B.			Per cent.
Up to 175..	1.0	1.0	38.2	0.7	0.7	37.0	0.3		0.3
175 to 200..	1.4	2.4		1.1	1.8		.3		.6
200 to 225..	2.1	4.5		2.1	3.9		.0		.6
225 to 250..	8.4	7.9		3.3	7.2		.1		.7
250 to 275..	5.4	13.3	30.8	4.8	12.0	29.8	.6		1.3

ULTIMATE TEST.

OIL	Gravity at 60° F.	Viscosity 20° C.	Heating value per pound.	Water.	Sulphur.	Sand.	Flash point, Pensky-Martens tester, closed.	Burning point, Pensky-Martens tester, opened.	Remarks.
Fresh.....	*B. 14.3	*Engler. 625	B. t. u. 18,493	Per ct. 0.6	Per ct. 0.79	Per ct. None..	*F. 188	*F. 336	Very viscous at 32° F.
Stored 6 years...	14.0	737	18,522	.6	.86	..do...	214	318	Do.

RÉSUMÉ.

In summing up it is well to repeat that the best all-around container in use at the present time for storing oil is the all-steel tank of gas-tight construction.

Tanks that are used for the accumulation of fresh oils from the well should invariably be of this construction, and as a rule it will probably pay to equip them with water-seal tops, if not also with some form of tile encasing or lagging.

Other devices for lessening the temperature of the oil in the tanks that can be cheaply applied and economically maintained, such as sprinkling with water in hot weather and painting the tanks white, are worth while.

To store gasoline or light distillate in tanks that have not tight tops is the height of folly, and it is poor judgment not to use also some type of cooling device.

Large concrete-lined reservoirs, as at present constructed, should not be used for the storage of fresh oils or of light oils.

It will pay to line a reservoir with concrete even though heavy oil only is to be stored.

In most cases it would probably also pay to put a concrete roof on the reservoir and cover it with earth—at least such a type of structure is worthy of consideration, no matter what the gravity of the oils to be handled.

Concrete, if properly proportioned, mixed, poured, tamped, and floated, can be made impervious to heavy oils without the addition of so-called "oil-proofing" compounds.

Contrary to popular engineering opinion, expansion joints are not necessary in properly constructed concrete linings for oil reservoirs in temperate climates, and no injury will result to the linings from their omission if the reservoirs be kept reasonably full of oil, or if, when the tanks are not in use, they are kept partly filled with water.

If crude can be refined at any profit, it should be put through the refinery as soon as possible after it is taken from the wells. If refining will not pay, the period of storage should be as short as possible because, so far as the oil is concerned, at best, each day of storage will entail a loss.

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FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

THE DIESEL ENGINE

ITS FUELS AND ITS USES

BY

HERBERT HAAS



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THE DIESEL ENGINE; ITS FUELS AND ITS USES.

By **HERBERT HAAS.**

INTRODUCTION.

The Bureau of Mines is endeavoring to reduce waste and increase efficiency in the production, refining, and utilization of petroleum. During the last few years the demand for petroleum and its products has increased so much more rapidly than the production that greater efficiency in utilization is now a matter of most serious importance.

The Diesel engine, a comparatively new type of internal-combustion engine, is an important device for insuring more efficient utilization of petroleum and coal-tar products, for it consumes heavy liquid fuels such as can not be utilized in other types. Like the gas or the gasoline engine, it is revolutionary within its province. For some uses this engine is replacing gas and gasoline engines, but its most important duty will be to replace the wasteful consumption of fuel oil in steam engines. Not only will a wider use of the Diesel engine relieve the demand, but it will result in more power being obtained from the same consumption of fuel oils. In spite of these advantages, comparatively few Diesel engines are in use in the United States.

In this report the author discusses recent developments in the design and construction of the Diesel engine, the fuels suitable for burning in it, and the uses to which it is particularly adapted.

GENERAL CHARACTERISTICS AND TYPES OF OIL ENGINES.

The term "oil engine," as generally used at the present time, is applied to internal-combustion engines that burn directly in the cylinder heavy liquid fuels of high boiling points, the fuel being injected into the compressed air shortly before or at the completion of the compression stroke. The distinguishing features of oil engines are that the fuel vapor is not absorbed by air before it is admitted to the cylinder, and that no inflammable mixture of vapor and air is compressed preceding its ignition. Oil engines compress air alone, and the heat of compression is used to ignite the fuel, which burns by consuming the oxygen of the air in the cylinder, the engine transforming the heat energy into work.

To facilitate and accelerate the burning of a liquid fuel it must either be vaporized, atomized, or intimately mixed with air immediately preceding its ignition.

Light, highly volatile liquid fuels, such as benzols, gasolines, alcohol, and distillates, offer no particular difficulties to vaporization; the air in its passage to the engine cylinder readily absorbs the fuel vapors and forms a combustible mixture, which is ignited electrically in the cylinder.

The process of charging the air with fuel vapors is called carburation. The more volatile fuels, like gasoline, can be carbureted at ordinary atmospheric temperatures; that is, without previous heating of the air or the fuel. Liquid fuels with higher boiling points may require heating of the air or the fuel, or both, to bring about their evaporation and absorption by the air preceding combustion. In the heavy-oil engine the vaporizing of the fuel takes place inside of the engine. As a fuel with a high boiling point can not be evaporated at moderate temperatures, thorough mechanical division preceding ignition and combustion is necessary.

THREE GENERAL TYPES OF LIQUID-FUEL ENGINES.

According to the means used for atomizing the liquid fuels and igniting them, there are two mechanically and thermodynamically distinct types of engines. In one type the entire fuel charge is sprayed against a highly heated surface in a chamber connected with the working cylinder. Contact with this highly heated surface gasifies the fuel, which is ignited and burns with explosion-like rapidity. Engines of this type are properly termed "explosion oil engines," or engines in which the fuel is burned at constant volume.

In engines of the other type the fuel to be converted is finely subdivided by air, and in this act of atomization is injected directly into the engine cylinder, where it is ignited automatically by the highly heated air in the cylinder. The combustion is not explosion-like, but is prolonged at constant pressure for the entire period during which the fuel is injected into the cylinder. This type of engine is universally known as the Diesel engine, being named after the late Rudolph Diesel, of Munich, Germany, its inventor. It is also termed a "constant-pressure oil engine."

There is a third general type of engine, combining features of the two types mentioned, in which the fuel is burned at both constant volume and constant pressure. Engines of this type are known as Sabathé engines.

According to whether an engine receives a working impulse every other revolution or each revolution, it is said to have a four-stroke cycle or a two-stroke cycle. Either type of engine may be single acting or double acting, such constructions being wholly mechanical and influencing in no manner the thermodynamic cycle of an engine.

All the three general types of oil engines enumerated may be built to work either on the four-stroke cycle or the two-stroke cycle and may be either single or double acting.

To summarize, differentiation must be made between the following types of oil engines:

1. Explosion oil engines.
 - (a) Engines having a four-stroke cycle.
 - (b) Engines having a two-stroke cycle.
2. Diesel engines.
 - (a) Engines having a four-stroke cycle.
 - (b) Engines having a two-stroke cycle.
3. Sabathé engines.
 - (a) Engines having a four-stroke cycle.
 - (b) Engines having a two-stroke cycle.

EXPLOSION OIL ENGINES.

ENGINES HAVING A FOUR-STROKE CYCLE.

Figure 1 shows the elements of an explosion oil engine having a four-stroke cycle. During the first stroke of the piston downward, which creates a partial vacuum in the cylinder, atmospheric air is admitted through the air-admission valve *a* into the cylinder. On the return stroke of the piston valve *a* is closed, and the air in the cylinder is compressed. Before the piston is at the inner (upper) dead center the fuel charge is sprayed by the fuel pump *b* into the chamber *c*, known as a hot bulb or hot ball, which is kept at a dull, cherry-red heat. As the piston reaches its inner (upper) center, the fuel is burned with explosion-like rapidity, the temperature and pressure in the cylinder increase, and the piston receives a power impulse, the gaseous mixture expanding during the second downward stroke of the piston. On its return stroke the exhaust valve *d* is opened and the piston sweeps the gaseous products before it and through the valve *d*. The working cycle of four strokes is then repeated.

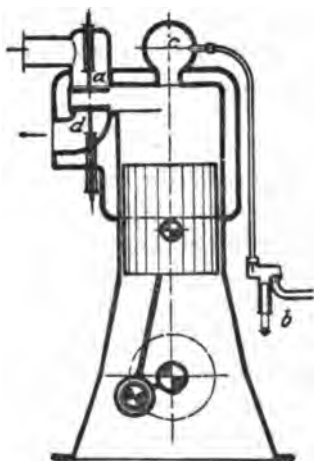


FIGURE 1.—Explosion oil engine having a four-stroke cycle. *a*, air valve; *b*, fuel pump; *c*, hot bulb; *d*, exhaust valve.

Before the engine is started the chamber or bulb *c* must be heated with a torch to a dark-red color, after which the successive explosions of the fuel oil will keep it hot. Inserted into this chamber in some engines is a thin-walled copper pipe against which the oil is sprayed. This small pipe can be heated quickly and readily absorbs heat from the chamber. It therefore tends to reduce the time required for starting the engine, which can thus start before the entire bulb is heated, and it also reduces missed ignitions after the

engine is operating. Heating of the bulb sufficiently to permit starting the engine requires 10 to 20 minutes. Compressed air, usually furnished by a small independent air compressor, operated by a gasoline engine, is used to start the engine.

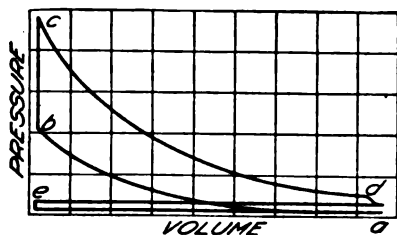


FIGURE 2.—Theoretical pressure-volume diagram of the operation of an explosion oil engine having a four-stroke cycle.

Figure 2 shows a theoretical pressure-volume diagram from an engine of this type. The air in the cylinder is compressed from *a* to *b*; at *b* the fuel is admitted and ignited and burns at constant volume from *b* to *c*. The line from *c* to *d* represents expansion, under which the piston moves forward, and at *d* the exhaust valve is opened, with the pressure dropping nearly to that of the atmosphere; the line from *d* to *e* represents the pressure-volume relation during the period in which the piston expels the products of combustion. The line from *e* to *a* shows a similar relation during the period in which the piston draws in a fresh supply of air, the pressure here represented being slightly below atmospheric pressure.

ENGINES HAVING A TWO-STROKE CYCLE.

Figure 3 illustrates the working of an explosion oil engine having a two-stroke cycle. Engines of this type have a closed crank case, provided on one side with large disk valves opening inwardly to the crank case and serving as air intakes. A large conduit connects the crank case with the cylinder, the conduit ports being so placed in the lower part of the cylinder that they are closed by the piston during the greater part of its travel; they are uncovered only a few degrees before it reaches its lower dead center.

Figure 3, A, shows the engine with a crank case full of air compressed by the previous downward travel of the piston, this internal pressure having closed the disk valve *a* leading to the atmosphere. The piston has traveled to its

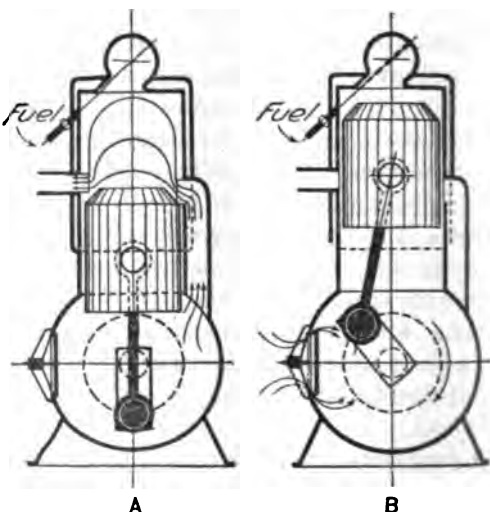


FIGURE 3.—Explosion oil engine having a two-stroke cycle. A, view when piston is at outer (lower) dead center; B, view when piston is approaching inner (upper) dead center.

outer (lower) dead center and uncovered the exhaust port *b* and the air port *c*. The pressure in the cylinder is higher than that of the atmosphere, so that the burned gases are swept out of the exhaust-gas port. The air port *c* admits fresh compressed air into the cylinder, sweeping it free of the products of combustion and refilling it with air needed for burning the next fuel charge. On its upward travel the piston covers the exhaust and air ports, compressing the air thus imprisoned in the cylinder. The air still in the crank case expands and its pressure falls below that of the atmosphere. Consequently the disk valve opens automatically and lets in a fresh supply of air (fig. 3, B). Just before the piston reaches its inner (upper) dead point a fuel charge is sprayed against the heated surface of the hot ball; the vaporized fuel is ignited and burned with a great increase of temperature and pressure by the time the piston has passed its inner dead point. The resulting impulse drives the piston downward, the two-stroke cycle being then repeated.

Figure 4 shows a pressure-volume diagram of such an engine. It is in principle the same as that of the engine having a four-stroke cycle, except that the exhausting of the gases, or scavenging, and the refilling of the cylinder with fresh air takes place along *dea*, *d* marking the point of opening of the exhaust ports several degrees before the outer dead point (at *e*) of the piston. Hence, only two strokes are required for a complete working cycle. These engines are started in the same manner as engines having a four-stroke cycle.

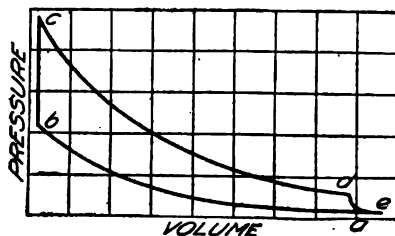


FIGURE 4.—Theoretical pressure-volume diagram of an explosion oil engine having a two-stroke cycle.

In explosion oil engines, the air is compressed in the cylinder to a pressure of 145 to 220 pounds per square inch and in the crank case to a pressure of 1 to 7 pounds per square inch. The explosion pressure varies from 270 to 470 pounds per square inch. The cylinder and the cylinder head are water-cooled, with the exception of the hot ball, which is kept at a dull red heat. The construction described is the one generally accepted and is to be preferred on account of its greater simplicity. Engines of this type are built both horizontally and vertically, the latter construction being preferred in multicylinder units.

The air for scavenging could also be furnished by constructing the working piston as a differential piston, traveling with the enlarged part in an air cylinder, or by an independent air pump. This design would result in a more complicated and expensive engine, hardly justified by its limitations.

DIESEL ENGINES.

ENGINES HAVING A FOUR-STROKE CYCLE.

Figure 5 shows the elements of a Diesel engine having a four-stroke cycle. Important adjuncts of the engine comprise a two-stage air compressor *a*, air-injection bottle *b*, compressed-air container *c*, and fuel-oil tank *d*.

The engine piston has started on its downward travel. The air-admission valve *e* is open to the atmosphere and lets air into the engine cylinder. On its return stroke the valve *e* and all other valves are closed, so that the piston compresses the air in the cylinder to a pressure of 450 to 500 pounds per square inch. As a result the air becomes highly heated, its temperature rising to about 1,000° F. when the highest pressure is reached. When the piston is at its upper dead center, the fuel-injection valve *f* is opened and liquid fuel in a volume proportioned to the load of the engine is forced into the cylinder, where, meeting the highly heated air, it automatically ignites and burns. When an automatically regulated supply of fuel has been delivered, valve *f* closes. Under the impulse of the expanding gases, the piston moves downward, transforming the heat energy of the fuel into work. Arrived at its lower dead center, the piston reverses its travel and begins a new upward stroke. The exhaust valve *g* being open, the piston sweeps the products of combustion before it, expelling them through this valve into the atmosphere, thus completing the four-stroke cycle. The cycle is repeated when air valve *e* opens again and the piston starts on its downward stroke.

For the sake of simplicity in describing the main features of the engine, no mention has so far been made of the important adjuncts of the fuel pump, the air compressor, and the fuel injector.

Pump *h* is under the influence of the engine governor, by which the volume of fuel delivered is proportioned to the load of the engine. The fuel oil stored in tank *d* flows by gravity to the pump, which delivers a measured quantity of oil to the fuel injector *f* during the suction stroke of the engine piston, while the needle valve of the injector is closed. The fuel injector is connected with the small air receiver *b*, in which air under a pressure of 700 to 900 pounds per square inch is stored, and air at this pressure always fills the valve chamber around the valve stem of the fuel injector.

Pump *h* in delivering its measured quantity of oil fuel to the injector must overcome the air pressure within the valve chamber. The pump is, therefore, designed to deliver the oil at a pressure of 100 to 200 pounds per square inch higher than the pressure of the air from the tank *b*, or at a pressure of 800 to 1,100 pounds per square inch. The fuel oil is delivered near the bottom of the fuel injector, immedi-

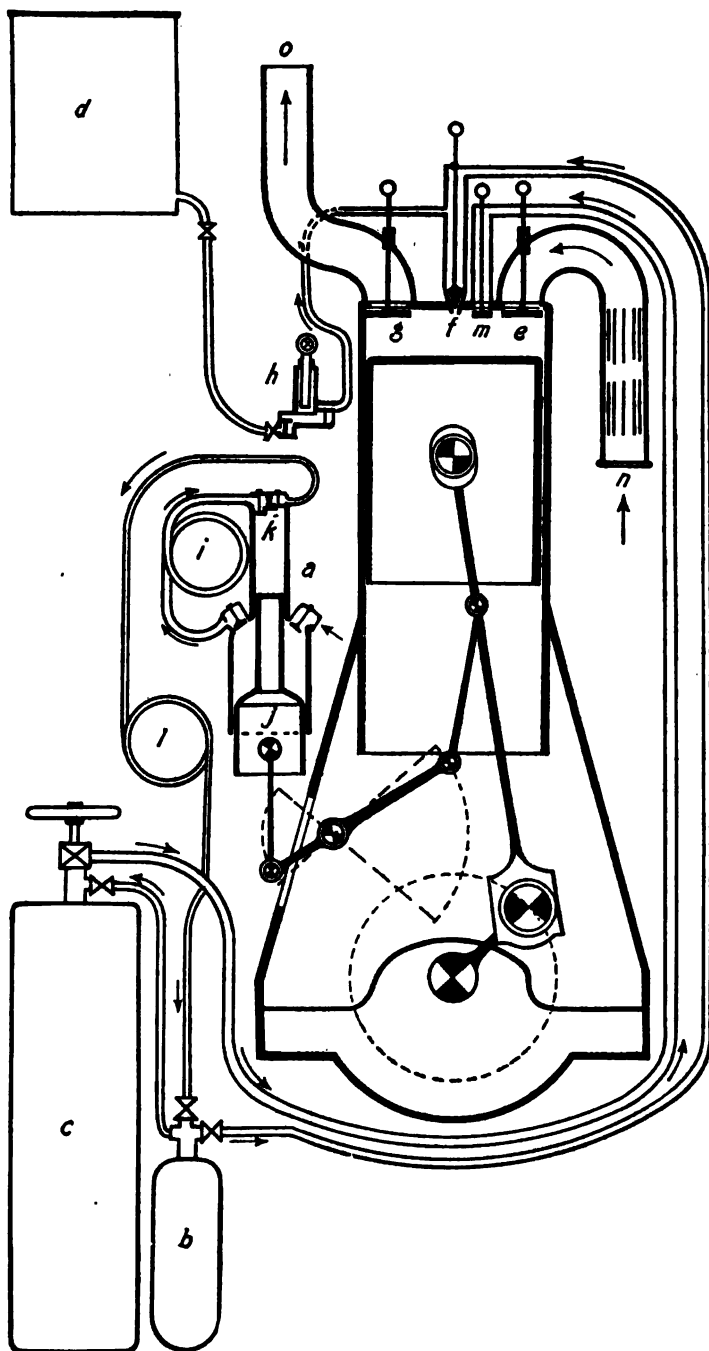


FIGURE 5.—Diesel engine having a four-stroke cycle. *a*, air compressor; *b*, injection air bottle; *c*, reserve air receiver; *d*, fuel tank; *e*, air (intake) valves; *f*, fuel-injection valve; *g*, exhaust valve; *h*, fuel pump; *i*, intercooler; *j*, low-pressure cylinder; *k*, high-pressure cylinder; *l*, aftercooler; *m*, starting valve; *n*, air intake; *o*, exhaust.

ately above the needle valve. When this needle valve is opened the injection air, which is at a pressure nearly double that of the compressed air in the engine cylinder at the completion of the compression stroke, atomizes the fuel oil and carries it into the engine cylinder. The injection air thus serves two important functions, namely, to inject the fuel into the cylinder and to subdivide it finely. The latter function is by far the more important, as on its efficiency greatly depends the success with which the heavy liquid fuels are burned. The fuel could be injected by direct pump pressure, but unless thoroughly atomized by highly compressed air it would burn only partly and after-ignition would result. This subject is treated more fully under the discussion of fuel injection.

The two-stage air compressor *a* furnishes the injection air; *i* is an intercooler to cool the air from the low-pressure cylinder *j* to atmospheric temperature before delivering it to the high-pressure cylinder *k*; *l* is an aftercooler to deliver cold air to tanks *b* and *c*. As previously mentioned, in the compressor the atmospheric air is brought to a pressure of 700 to 1,000 pounds per square inch. The air receiver *b* provides reserve air storage. Air is drawn from it whenever the engine is started and is delivered through the starting valve *m*, when the piston is at the upper dead center, in position to receive a power impulse, air being the power medium for starting the engine. After the engine has been started, tank *c* is recharged from tank *b*, by working the air compressor at its full capacity for 15 to 20 minutes. When the desired air pressure has been restored in tank *c*, all valves leading to and from the tank are closed. The air-inlet pipe is shown at *n* and the exhaust pipe at *o*.

Starting a Diesel engine from no load to full load requires one to three minutes, depending on the size and the construction of the engine.

ENGINES HAVING A TWO-STROKE CYCLE.

Figure 6 shows the elements of a Diesel engine with a two-stroke cycle. It differs from the engine just described in that it has a scavenging air pump *a* and exhaust ports *b*, arranged around the cylinder walls, covered and uncovered by the piston in its upward and downward travel, and two or more air-admission valves in the cylinder head.

The scavenging air pump *a* is double acting and its air delivery is controlled mechanically by a piston valve. The air admission to the engine cylinder is mechanically controlled by two or more air valves *c* in the cylinder head. The scavenging air is compressed to a pressure of 3 to 7 pounds per square inch.

The air is admitted into the working cylinder as soon as the exhaust ports have been uncovered by the piston; the cylinder, full of the gaseous products of combustion, is cleared by the scavenging air,

which sweeps out through the exhaust ports *b*, leaving the cylinder full of a new charge of fresh air. As the piston on its return stroke closes all cylinder valves and exhaust ports, the air is compressed in the same manner as in the engine having a four-stroke cycle, and

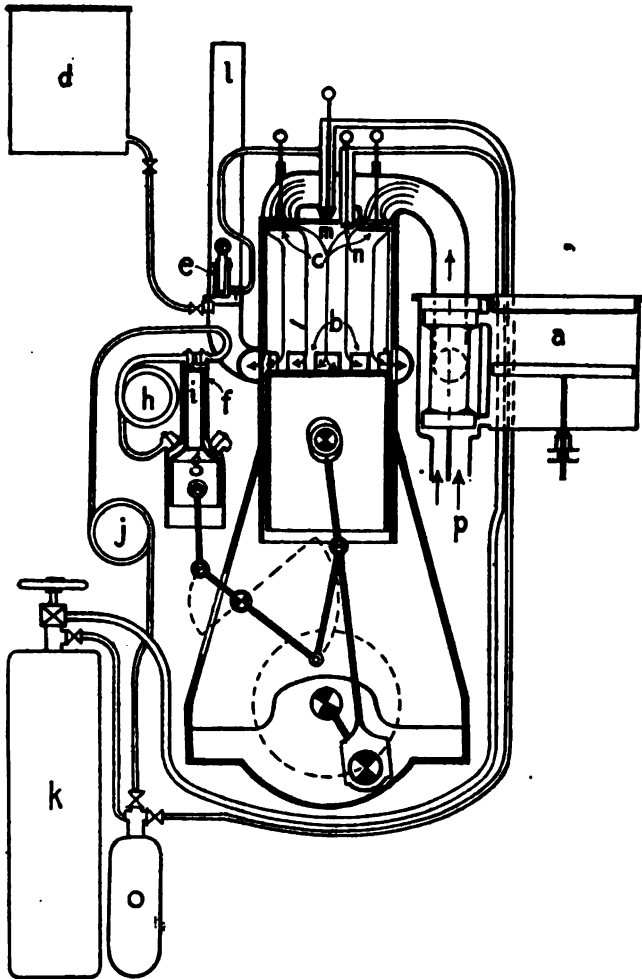


FIGURE 6.—Diesel engine having a two-stroke cycle. *a*, Pump for air for scavenging; *b*, exhaust ports; *c*, air valves; *d*, fuel tank; *e*, fuel pump; *f*, air compressor; *g*, low-pressure cylinder; *h*, intercooler; *i*, high-pressure cylinder; *j*, aftercooler; *k*, air receiver; *l*, exhaust pipe; *m*, fuel injection valve; *n*, starting air valve; *o*, injection air bottle; *p*, air inlet pipe.

the fuel charge is injected on the completion of the compression stroke.

During the next stroke the expanding gases give the piston its power impulse; with the opening of the exhaust ports and the discharge of the spent fuel gases, the working cycle is complete. By employing a scavenging air pump for cleaning the cylinder of its com-

bustion products and refilling it with fresh air, a working impulse to every revolution is obtained, whereas two complete revolutions to a working impulse are required in an engine having a four-stroke cycle.

In figure 6 the air-admission valves are shown in the cylinder head of the engine. In some later engines these scavenging air valves

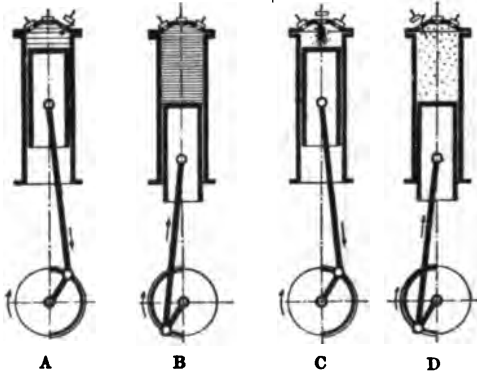


FIGURE 7.—Working diagram of single-acting Diesel engine having a four-stroke cycle. A, intake; B, compression; C, working stroke; D, exhaust.

have been placed in the cylinder wall, opposite the exhaust ports, somewhat similar to the placing shown in figure 3 for the explosion oil engine having a two-stroke cycle. The piston is shaped so as to direct the travel of the scavenging air in the cylinder and to arrange it in currents that sweep the cylinder clean of the products of combustion. Thus the cylinder head is relieved of numerous air

valves, which, by reason of space limitations imposed by the cylinder head, have to be of restricted area, necessitating greatly increased air velocities to effect the desired scavenging. The placing of these valves in the cylinder head also results in a complicated head casting, the water spaces of which are narrow and restricted, so that effective cooling is made more difficult. By the modified construction above mentioned, the cylinder head has only starting and fuel injection valves. The placing of the air valves in the cylinder head has the advantage of insuring a more thorough scavenging, as the air, entering at the top of the cylinder, will more effectively sweep before it the burned gases. The working cycles of the two engines are illustrated in figures 7 and 8; the corresponding indicator diagrams are shown in figures 9 and 10. The illustrations are taken from an article by Diesel.^a

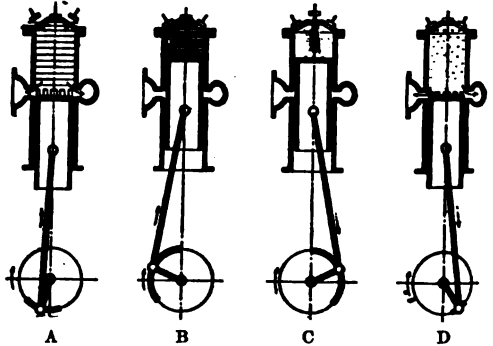


FIGURE 8.—Working diagram of single-acting Diesel engine having a two-stroke cycle. A, scavenging; B, compression; C, working stroke; D, exhaust.

^a Diesel, R., *The Diesel oil engine: Engineering* (London), vol. 93, 1912, p. 395.

SABATHÉ ENGINES.

Sabathé engines differ from the Diesel engines described only in the method of admitting fuel and that of timing ignition, these being accomplished by a fuel-injection valve and a governor of a construction peculiar to these engines. A part of the fuel is admitted before the upper dead center of the piston is reached, and burns with constant volume. The remainder is admitted at the completion and return of the stroke, and burns at constant pressure. If the engine load diminishes, the fuel admitted during the second, or constant-pressure, period is diminished, stopping entirely during low load when the only fuel admitted is burned at constant volume.

The peculiarity of the working cycle of these engines is well illustrated in the pressure-volume diagram shown in figure 11. The burning of the fuel is accomplished at constant volume as represented by the line bc , and at constant pressure as represented by the line cc' . The fuel, of course, is not introduced intermittently at b and c , but is admitted continuously, the burning at constant volume and pressure being controlled by the action and timing of the needle valve, under the influence of the governor, at predetermined positions of

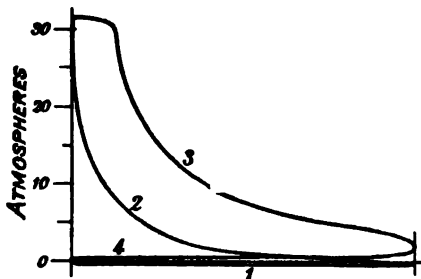


FIGURE 9.—Indicator diagram of single-acting Diesel engine having a four-stroke cycle. 1, intake; 2, compression; 3, expansion; 4, exhaust.

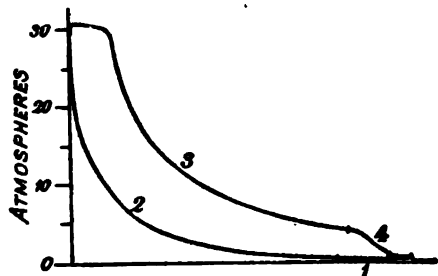


FIGURE 10.—Indicator diagram of single-acting Diesel engine having a two-stroke cycle. 1, scavenging; 2, compression; 3, expansion; 4, exhaust.

that it burns at constant volume. Line bc therefore inclines slightly toward the expansion line. The properties of the fuel, whether it burns readily or slowly, the temperature in the cylinder, and the means used for its ignition all influence the burning process. A part of the fuel usually does not burn until expansion has begun.

Likewise, in Diesel engine diagrams, there is no perfect constant-pressure line. With the admission of the fuel, there is a slight increase

of the piston. Line cc' varies in magnitude with the load. At low load, line cc' disappears entirely; then the diagram becomes similar to that of an ordinary explosion oil engine. The diagram areas are proportionately decreased.

Actual diagrams differ from the theoretical one shown in figure 11. In reality it is not possible to burn the fuel so quickly

in pressure; there is also an after-burning of incompletely burned fuel when expansion has started, so that the initial and the terminal parts of the constant-pressure line are slightly curved.

In explosion oil engines the constant-volume line bc of figure 11 is shortened, and the diagram area is decreased, with a decrease in

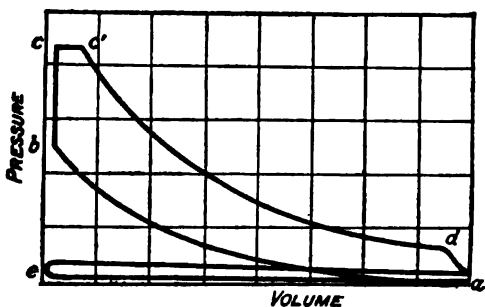


FIGURE 11.—Pressure-volume diagram, Sabathé four-stroke cycle engine.

the fuel supply during fractional loading of the engine.

In the Diesel engine the constant-pressure line is shortened, and the diagram area decreased, with partial loads. In some two-stroke engines that increase the specific duty by increasing the scavenging air pressure; that is, by pumping a greater weight of air into the work-

ing cylinder, the compression pressure is variable, and decreases with a decrease in load to the minimum required for satisfactory burning of the fuel. Figure 12 shows actual indicator diagrams for different loads of a Diesel engine having a four-stroke cycle.

COMPARATIVE ADVANTAGES OF FOUR-STROKE AND OF TWO-STROKE CYCLES.

As regards the relative theoretical advantages of the four-stroke and the two-stroke engines, the two-stroke engine appears to be superior. For a given cylinder size and speed, the specific duty of an engine having a two-stroke cycle is double that of an engine having a four-stroke cycle, for it receives a power impulse for every revolution as against one for every two revolutions of the engine having a four-stroke cycle. Its mechanical efficiency also would appear to be greater, as the engine with a four-stroke cycle has to make two strokes for expelling the products of combustion and for filling the cylinder with a fresh supply of air for burning the next fuel charge. The power for this work has to be drawn from the flywheel, in which it has to be stored again during the power stroke. As in addition the cyclic regularity of an engine with a four-stroke cycle but having the same power and speed is greatly less, a much heavier flywheel is required. The four-stroke engine is also much heavier per horsepower than the two-stroke engine.

In practice, however, these advantages are not so apparent. Engines having a two-stroke cycle require an auxiliary air pump for performing the work of scavenging and of refilling the working cylinder with fresh air, work that in the engine having a four-stroke cycle is performed by the engine directly in the working cylinder. The air

pump, proportioned to the engine it has to supply, is a large and heavy piece of machinery, requiring considerable power for its operation. To insure thorough scavenging of the working cylinder, a volume of fresh air greater than the volume of the cylinder must be supplied by the air pump. The pump is usually proportioned to supply one and three-tenths times the working cylinder volume of air. Even then sweeping of the working cylinder clean of the remnants of the products of combustion is difficult, and any gases remaining in the cylinder displace a proportionate amount of oxygen needed to maintain combustion, thus lowering the possible power output of the engine.

The heat exchange in an engine having a two-stroke cycle being more rapid than in one having a four-stroke cycle, the material of the former is more severely taxed, and it is not advisable to use as high mean effective pressures nor piston speeds as in engines having a four-stroke cycle. High mean effective pressure results in high terminal pressure at the time of exhausting, with a consequent loss in power. As the exhaust ports of two-stroke engines are uncovered by the piston,

and as exhausting of the gases must be rapid, the piston has to uncover these ports several degrees before it reaches its lower dead center. High mean effective pressure will therefore cause proportionately greater losses in an engine having a two-stroke cycle than in one having a four-stroke cycle. These features combine to diminish the theoretical advantages that engines having a two-stroke cycle appear to possess over engines with the simpler four-stroke cycle.

To avoid any misconception it is well to emphasize that the difficulties and the considerable fuel losses during exhausting experienced

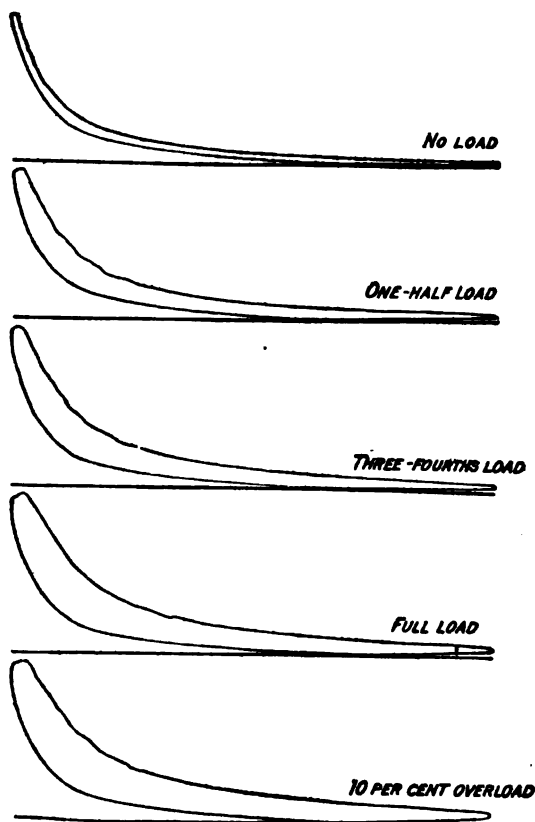


FIGURE 12.—Indicator diagrams of a Diesel engine having a four-stroke cycle.

with the usual two-stroke gas engine, chiefly on account of the successive scavenging and loading of the engine cylinder, are absent in the two-stroke Diesel engine. In this engine the same air pump does the scavenging and the charging, whereas in the two-stroke gas engine separate pumps are used for scavenging and for charging the cylinder with the gas-and-air mixture. These pumps consume considerable power, and it is difficult to prevent a part of the fresh gas-air charge from being expelled with the products of combustion during exhausting. Having to deal with inert air (air not mixed with any combustible gas) and a liquid fuel, the working process of the two-stroke-cycle Diesel engine is much simpler than that of the gas engine having a two-stroke cycle.

Engines with a four-stroke cycle, in sizes from 50 to 800 horsepower,^a have been developed to a high degree of perfection, and are so simple in operation that little justification for the adoption of two-stroke engines in these sizes exists at present. The field of two-stroke types lies chiefly in engines of very small and very large powers, the latter from 1,000 horsepower upward.

ESSENTIAL QUALITIES OF ENGINES OF SMALL POWER.

Engines of small power should combine lightness with the greatest simplicity in design, to assure low manufacturing costs and ease of operation in inexperienced hands. In low-powered engines fuel economy and refinements in design are sacrificed to low first cost of engine. Nor is fuel economy of so much moment in small engines, as the total fuel consumption is a relatively small source of expense. Explosion engines having a two-stroke cycle best meet these requirements. Engines of this type have merely self-acting disk valves for admitting air to the crank case of the engine, and a fuel pump delivering a measured amount of fuel to the fuel-injection nozzle. This construction insures simplicity, but also results in low volumetric and mechanical efficiency and low fuel economy. Diesel engines, to insure high fuel economy, demand mechanically operated valves, and a high-pressure air compressor, which, together with the high compression and resulting high temperatures used, demand the highest grade materials and workmanship, involving a high cost per horsepower.

Although in European countries Diesel engines are manufactured in sizes as small as 15 horsepower there is little demand for this size in America, where fuel prices generally range lower, and where Diesel engines of 75 horsepower indicate the probable commercial limit in size. For large powers, from 1,000 horsepower upward, engines having a two-stroke cycle are preferred, as in these sizes limitations are put on the size of cylinder practical for engines having a four-stroke cycle by the high temperatures and pressures produced in such engines.

^a The British horsepower is used throughout this publication. The British horsepower is 1.0139 metric horsepower, or 76.041 m. kg., whereas the metric is 75 m. kg. per second.

An increased use of material to withstand the greater total pressure in large cylinders merely accentuates the difficulty of cooling the cylinder, the cylinder head, and the piston, and of carrying off the heat fast enough through thick cylinder and cylinder-head walls. Stresses due to unequal expansion and contraction may easily lead to ruptures of vital engine parts, such as cylinder heads, pistons, and cylinders. Successful building of large Diesel engines must therefore be fortified by a great amount of practical experience, particularly in the rational design and selection of casting mixtures with correct chemical and physical properties.

Twenty-four inches in cylinder diameter represents the probable upper limit with air-cooled pistons, and 30 inches with water-cooled pistons, for Diesel engines having a four-stroke cycle.

COMPARATIVE ECONOMIES OF DIESEL AND OF EXPLOSION OIL ENGINES.

The use of explosion oil engines should be dictated entirely by their over-all economy. Although they are materially cheaper in first cost, they consume considerably more fuel and lubricating oil than Diesel engines, and their fuel consumption at fractional loads increases at a greater rate than does that of Diesel engines.

The difference in economy between explosion oil engines and Diesel engines is due not so much to a difference in thermodynamic cycles as in constructional differences, which decidedly favor the Diesel engines.

The higher the initial temperature and the lower the terminal temperature the more perfect will be the heat utilization. High initial temperature is a function of pressure. If the compression were carried as high in the explosion engine as in the Diesel, the explosion oil engine theoretically would show a slightly higher thermodynamic efficiency than the Diesel engine. The final pressure, when the fuel is burned at constant volume, would, however, be greatly increased above the compression pressure, or to about 60 atmospheres, with an accompanying rise in temperature far beyond that practicable with the materials of construction available. These limitations impose a lower compression pressure on explosion oil engines than on Diesel engines, so as to keep the maximum pressure and temperature within safe limits of permissible engine construction.

Thus with a compression pressure of 150 to 250 pounds per square inch the explosion pressure becomes 270 to 500 pounds per square inch; and with the instant ignition and burning of the previously vaporized oil common to explosion oil engines, a temperature of 2,300° to 3,150° F. is reached.

The Diesel engine has a higher but more gradually increasing compression pressure (450 to 500 pounds per square inch) which does not subject the engine to sudden shocks, with a resulting increase in temperature from atmospheric to about 1,000° F. The fuel is gradual-

ly injected as the piston moves from its top center (inner dead center) downward, the pressure remaining practically constant during the time of fuel admission. With a decrease in load, the time of fuel admission is also shortened, that is, the fuel supply is shut off sooner. Therefore the increase in temperature due to the burning of the fuel at constant pressure does not exceed that reached in an explosion engine, notwithstanding the higher compression pressure used in the Diesel engine, and the higher initial temperature caused by this compression. Thus, the temperature in a Diesel engine seldom exceeds 2,600° F., and reaches 3,000° F. only when the engine is overloaded.

If, then, the working cycles of the two types of engines are compared on the basis of compression pressures used, the Diesel engine is found to have a greater thermal efficiency, because it can work successfully with the higher compression pressure. This superiority is confirmed by comparative entropy diagrams.

Whereas in explosion oil engines the efficiency is influenced by the compression ratio alone, in the Diesel engine the efficiency is influenced by the compression and the cut-off ratios, the efficiency increasing with a decrease in the length of the cut-off or constant-pressure line. Thus, at fractional loads, the indicated thermal efficiency of Diesel engines increases, which partly offsets the loss in mechanical efficiency, that is, the increased fuel consumption for performing the internal work of the engine. This accounts for the very "flat" fuel-consumption curve of Diesel engines, which maintain an almost constant fuel economy over a fairly wide range in load. Thus at three-fourths load the increase in fuel consumption per brake horsepower-hour is only 2 to 5 per cent, and at one-half load 10 to 15 per cent greater than at full load in high-grade engines, which is in marked contrast with fuel increases in other prime movers. The ability to create higher initial temperatures, aside from increased thermal efficiency, enables the Diesel engine to burn a greater variety of fuels. In addition to the fuel being thoroughly atomized by highly compressed air, the heated oxygen has an augmented power of combining with the carbon and hydrogen in the fuel, so that the velocity of the chemical reaction at the high temperature in a Diesel engine is greatly increased. As a result the range of fuels suitable for the Diesel engine comprises such heavy liquid fuels as petroleum residues, coal-tar oils, and coal tars.

A serious fuel loss in explosion engines is frequently caused by the decomposition of the fuel oil sprayed into the hot ball. Various hydrocarbons are formed with a separation of carbon and oil soot; part of this coats the hot ball and the cylinder, and a larger part is expelled with the gaseous products of combustion. From time to time accumulated soot in the exhaust piping catches fire and burns, necessitating precautions against fire from this source.

Comparative economies of Diesel and of explosion oil engines are shown in Table 1 following. On a basis of oil costing \$1 a barrel,

TABLE 1.—Comparative economies of Diesel and of explosion oil engines.

[Cost of fuel oil taken as at \$1 a barrel of 320 pounds; cost of lubricating oil taken as at 35 cents a gallon.]

Item.	Diesel engine, European make. ^a						Diesel engine, American make. ^a						Explosion oil engine, European make. ^b						Explosion oil engine, American make. ^b					
	Load.	Fuel per horsepower.	Per cent.	Barrels per year.	Fuel cost per horsepower - year (\$.760)		Load.	Fuel per horsepower.	Per cent.	Barrels per year.	Fuel cost per horsepower - year (\$.760)		Load.	Fuel per horsepower.	Per cent.	Barrels per year.	Fuel cost per horsepower - year (\$.760)		Load.	Fuel per horsepower.	Per cent.	Barrels per year.	Fuel cost per horsepower - year (\$.760)	
Fuel consumption or cost at different loads.....	$\left\{ \begin{array}{l} 4/4 \\ 3/4 \\ 2/4 \end{array} \right.$	$\left\{ \begin{array}{l} .43 \\ .44 \\ .50 \end{array} \right.$	$\left\{ \begin{array}{l} 100 \\ 104.8 \\ 119 \end{array} \right.$	$\left\{ \begin{array}{l} 883 \\ 678 \\ 686 \end{array} \right.$	$\left\{ \begin{array}{l} \$11.50 \\ 12.06 \\ 13.70 \end{array} \right.$		$\left\{ \begin{array}{l} 4/4 \\ 3/4 \\ 2/4 \end{array} \right.$	$\left\{ \begin{array}{l} .48 \\ .50 \\ .58 \end{array} \right.$	$\left\{ \begin{array}{l} 100 \\ 104.2 \\ 120.8 \end{array} \right.$	$\left\{ \begin{array}{l} 988 \\ 770 \\ 610 \end{array} \right.$	$\left\{ \begin{array}{l} \$12.15 \\ 12.68 \\ 16.27 \end{array} \right.$		$\left\{ \begin{array}{l} 4/4 \\ 3/4 \\ 2/4 \end{array} \right.$	$\left\{ \begin{array}{l} .60 \\ .68 \\ .76 \end{array} \right.$	$\left\{ \begin{array}{l} 100 \\ 108.8 \\ 125 \end{array} \right.$	$\left\{ \begin{array}{l} 1,232 \\ 1,013 \\ 770 \end{array} \right.$	$\left\{ \begin{array}{l} \$16.43 \\ 18.11 \\ 20.58 \end{array} \right.$		$\left\{ \begin{array}{l} 4/4 \\ 3/4 \\ 2/4 \end{array} \right.$	$\left\{ \begin{array}{l} .75 \\ .83 \\ .95 \end{array} \right.$	$\left\{ \begin{array}{l} 100 \\ 110.7 \\ 126.7 \end{array} \right.$	$\left\{ \begin{array}{l} 1,540 \\ 1,280 \\ 975 \end{array} \right.$	$\left\{ \begin{array}{l} \$20.53 \\ 22.76 \\ 26.00 \end{array} \right.$	
Oil consumed for lubrication, gallons per year.....	360 to 550.....						360 to 550.....						1,100 to 1,460.....						1,100 to 1,460.....					
Cost of lubrication.....	\$128 to \$193 per year, or \$1.70 to \$2.60 per horsepower-year.						\$128 to \$193 per year, or \$1.70 to \$2.60 per horsepower-year.						\$380 to \$510 per year, or \$5.10 to \$6.80 per horsepower-year.						\$380 to \$510 per year, or \$5.10 to \$6.80 per horsepower-year.					
Cost of installation, including building.....	\$90 per horsepower.....						\$90 per horsepower.....						\$60 per horsepower.....						\$60 per horsepower.....					

^a 75-horsepower, one-cylinder, single-acting engine having a four-stroke cycle.^b 75-horsepower, two-cylinder, single-acting engine having a two-stroke cycle.

the minimum yearly fuel cost in dollars corresponds to the number of barrels of oil consumed yearly. Oil usually costs more than \$1 a barrel, especially gas oils, which are the only oils suitable for use in explosion oil engines. On the other hand, residues and heavy fuel oils can be burned in the Diesel engine. Lubricating expense is also materially higher in explosion than in Diesel engines. With a fairly good load factor it will not take long to make up the difference in first cost by greater economy, particularly when the fuel cost is increased by long hauls.

DIESEL'S USE OF CARNOT CYCLE.

Diesel's original ideal heat engine was planned to use the Carnot cycle. Aside from the difficulty of obtaining isothermal changes with gaseous fuels of low specific heat, the original Diesel cycle called for entirely impractical pressures and temperatures. Functional and structural difficulties of his engine as originally planned necessitated many mechanical and thermodynamical changes before the first successful engine was developed, in 1895. The first commercial engine, of 60 horsepower, was built early in 1898, and is reported to be in successful operation still. The same year, three engines were exhibited at the Munich exposition, one built by the Augsburg Machine Works, one by the Deutz (Otto) gas engine works, and the third by Krupp. Sulzer Bros. of Winterthur, Switzerland, built their first Diesel engine, of 20 horsepower, in 1896-97. The commercial manufacture and distribution of Diesel engines dates from that time, and at present engines representing more than two million horsepower are in operation.

The Carnot cycle may be briefly described by reference to figure 13, in which T represents absolute temperature and S represents entropy. The line da represents isothermal compression during which the heat of compression is to be absorbed as it is added; from a to b adiabatic compression sets in, during which no heat is to be abstracted or added. During the isothermal expansion, from b to c , heat is added to maintain constant temperature, and from c to d the expansion is to take place adiabatically in such a manner that the relation of the various compressions and expansions cause the final volume, pressure, and temperature to coincide with those at the beginning of the cycle.

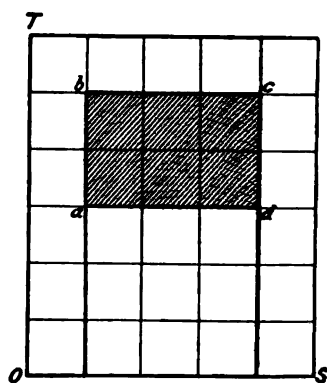


FIGURE 13.—Diagram explaining Carnot cycle.

This cycle demands that during the first part of the compression, heat must be perfectly carried away, that is, the working fluid and the surrounding material must be perfect conductors; and during the second period of the compression no heat must be carried away or added, thus calling for perfect nonconductors; similar opposing conditions are demanded during the expansion periods. Therefore it is not surprising that the theoretical requirement of a perfectly conducting and nonconducting material during different periods of the same cycle has never found practical fulfillment.

Diesel endeavored to meet the first condition by spraying water into the cylinder, the spray to be stopped during the adiabatic compression. During the isothermal expansion the fuel was to be injected into the cylinder to produce the required heat. The adiabatic compression had therefore to be high enough to heat the air to a point at which it would ignite the fuel. In the first Diesel engine planned, coal dust was to be the fuel. The adiabatic terminal pressure was to reach 250 atmospheres, and was to drop to 90 atmospheres with the termination of the isothermal combustion of the fuel. The cylinder was not to be water-cooled, but to be insulated against all radiation of heat. The thermal efficiency was to be 73 per cent. On account of the high pressure and temperatures involved and the small amount of useful work obtained, the original cycle was changed to that which has been in use for the past 20 years. It still remains the nearest approach to the Carnot cycle. The Diesel engine with its 45 per cent indicated thermal efficiency is the most efficient heat engine of to-day. The efficiency of the Diesel cycle is about 60 per cent, of which 75 per cent is realized in indicated work (45 per cent net indicated thermal efficiency) and 60 per cent in effective mechanical work (36 per cent net effective thermal efficiency) in high-grade, four-stroke engines at full load. Two-stroke engines have a slightly lower efficiency.

DETAILS OF DIESEL ENGINES.

Diesel engines are built in both vertical and horizontal units, each type showing some differences in construction in the frame, the valve gear, the fuel injector and the lubricating arrangements.

VERTICAL ENGINES.

Vertical engines are built as one cylinder and as multicylinder units. The structural details of the bedplate of such engines vary according to whether it is to support the closed crank case or individual cylinder frames, so-called A frames. For supporting the closed crank case the entire top joint of the bedplate is planed to insure an oil-tight joint between it and the crank case; for supporting individual cylinder frames the bed is provided with carefully machined

pads or surfaces to which the A frames are bolted. The bed is cast integral with the crank pit and the seats to receive the main-shaft bearings. The pit is as a rule used as a reservoir for collecting the spilled and surplus lubricating oil. Either method of building the engine has its advantages, which are more those of manufacture than of any inherent superiority of one arrangement over the other. A bedplate for supporting a closed crank case is shown in Plate I, A.

ADVANTAGES OF INDIVIDUAL CYLINDER FRAMES.

The choice of individual cylinder frames enables the manufacturer to build with greater ease two, three, four, or even five and six cylinder units from a smaller number of standardized cylinders and valve gears. With an engine having a closed crank case this is not so readily and cheaply accomplished. Inasmuch as most engines are used for the generation of electric current with direct-connected generators, four-cylinder units are necessary to obtain the desired cyclic regularity for satisfactory generator performance in engines having a four-stroke cycle, especially when two or more units are to operate in parallel. Multicylinder units also permit a higher number of revolutions, by shortening the stroke without necessarily increasing the piston speed of these units. Such features combine to produce a compact, relatively light engine. The closed crank case is the construction favored for medium and high speed engines, with which forced-feed lubrication is used. There is no spilling and spattering of lubricating oil, and the engine and engine-room floor are readily kept clean. The forced-feed lubricating system, at first used only on high-speed units, has proved so satisfactory and so economical in the use of lubricants that it is used increasingly on medium-speed and low-speed engines. In Europe manufacturers have adopted this closed crank case construction for units up to 1,000 horsepower.

In larger engines, which require the use of A frames, the panels are connected with steel sheets provided with large doors between the individual frames, so as to obtain the advantages of a closed crank case and of forced-feed lubrication.

On the bedplate is also mounted the multistage air compressor, usually in a vertical position to preserve the symmetry of the engine.

CONSTRUCTION OF CYLINDERS AND CYLINDER FRAMES.

The cylinder frame receives the cylinder liner which has at the head end a heavy flange fitted to the cylinder frame. A registered joint is either placed wholly in the top of the flange of the cylinder liner or is divided over the top of the cylinder frame and the liner. Some constructors prefer the latter method, to insure a water-tight



A. BED PLATE OF 120-BRAKE-HORSEPOWER VERTICAL ENGINE WITH CLOSED CRANK CASE.

Housings for main bearings are being scraped to true seat for bearing shells by turning in the housings a cast-iron shell, ground to exact diameter, covered with "blue grease" used in scraping off high spots. Some of the shells and cups are shown.



B. CONSTRUCTION OF EXHAUST-VALVE LEVER PERMITTING EASY REMOVAL OF VALVE.

and a gas-tight joint. The cylinder head is provided with a corresponding ring that fits into the registered joint, soft-copper packing rings being inserted between the surfaces.

The top end of the cylinder frame must be so designed as to provide ample metal, as in it are secured the stud bolts that hold the cylinder head and are subject to heavy stress.

The cylinder liner is provided in the middle with a machined rib or ring to center the liner and also to give it lateral support against a similar ring cast integral with the cylinder frame. The bottom part of the liner has another ring provided either with a gland ring or a groove into which fits a circular rubber packing ring. Both methods are used to insure a water-tight joint between mantle and liner. The cylinder liner is fixed only at the head end, and is therefore free to expand in the other direction.

The use of separate castings for the mantle and for the cylinder liner is important, as the cylinder liner becomes much hotter than the mantle, so that the unequal stresses produced in the two are likely to fracture a cylinder if cast integral with the mantle. Separate liners can be designed and made of the most suitable material; in case of excessive wear or fracture, they can be cheaply and easily replaced. On account of the rapid alternations of exceedingly high and low temperatures, cylinder liners are preferably made of a special close-grained cast iron of high tensile strength—capable of withstanding a load of 40,000 pounds per square inch—and able to withstand severe shock tests.

For cylinder lubrication the cylinder frame and the liner are perforated in four places, and oil passages consisting of small copper pipes are inserted between the mantle and the liner, and bridge the water space. These oil passages are in the same plane in the middle part of the cylinder, being spaced 90° apart, and 45° removed from the axis of the main shaft. As the front and the back cylinder walls receive the principal pressure, the oil feeds in these cylinders are sometimes set closer together, so as to feed the oil more freely to the surfaces most taxed.

The length of the cylinder liner is determined by the length of the stroke and of the piston. The length of the piston is governed according to whether it is a trunk piston or whether it is guided by a crosshead. The piston may extend one-fifth of its length out of the cylinder. Cylinder frames and liners of engines having a two-stroke cycle differ from those of engines having a four-stroke cycle, previously described, in that the bottom part has exhaust ports and water-cooled passages for conducting the products of combustion into the exhaust pipes. In some engines the scavenging air ports are also placed in that part of the cylinder.

HORIZONTAL ENGINES.

The frames of horizontal Diesel engines do not differ materially from those of ordinary horizontal gas engines. They are built either as one or two cylinder frames. If four-cylinder units are desired, two-cylinder frames are united by placing the flywheel and the driving pulley or electric generator between the two frames. The cylinder mantle and the crank pit, with bearing supports, are cast integrally.

The frame is provided with machined pads, to which the air compressor is bolted. The compressor is usually driven from an overhanging crank pin at the end of the main shaft. According to whether the connecting rod is connected to a crosshead carried in a guide or whether the trunk piston, with its piston pin, serves as a modified crosshead guided by the cylinder, the cylinder frame is proportionately long or short. To collect the spent cylinder oil, a cast-iron rib extends vertically upward and across the frame, separating the crank pit from the forward part of the cylinder frame and preventing the bearing oil collected in the crank pit from becoming mixed with the used cylinder oil.

The cylinder liner is constructed as for vertical engines and is set into the frame in much the same manner. Cylinder frames for engines having a two-stroke cycle are built along similar lines as for vertical engines, or the cylinder and jacket mantle, with cored exhaust ports and water-cooled conduit for exhaust gases, are cast in one piece, with the cylinder liner extended so as to be slipped into and bolted to a supporting ring which is part of the main engine frame; the overhanging cylinder frame is then supported by a suitable footing or column. The engine frame is designed to receive the injection air compressor as well as the scavenging air pump.

CONSTRUCTION OF CRANK SHAFT.

The high pressures common to Diesel engines demand proportionately heavy crankshafts. These are forged from the solid, a ductile low-carbon steel being used. Some manufacturers use nickel-steel shafts. For large engines, or for an engine made of two half units, the shaft is composed of more than one piece, although the common practice for stationary engines is to use one-piece shafts, even if four cranks are used.

METHODS OF LUBRICATION.

When the shaft and the connecting rods are not drilled for forced-feed lubrication, it is usual to provide each crank with a centrifugal oiling ring to deliver the oil to the crank pins. For this purpose the crank pins are drilled, the drilled passages receiving the oil from the rings and delivering it to the pin brasses. Oil-drip rings turned out of the solid material of the shaft confine the oil to the main bearings and lead it to the oil reservoirs in the crank pit. On the shaft is

mounted the helical gear for engaging a like gear on the governor shaft and driving it, the gears dipping into the oil. A crank pin for driving the air compressor is part of the main shaft and is lubricated in the same manner as the other pins. The flywheel and the driving pulley are usually mounted on an enlarged part of the shaft, the extension of which is carried in an outboard bearing.

The main bearings for this type of shaft are either provided with an oiling ring or are constantly flooded with oil under pressure issuing from a pipe in the cover of each bearing. The surplus oil is collected in the crank pit and is filtered and cooled, and is returned to the bearing by a pressure oil pump. A circulating system of this kind insures positive lubrication of the main bearings, whereas oiling rings may possibly stick and through failure of lubrication cause the burning of a bearing.

Another system of lubrication, essential to high-speed and even medium-speed engines, and, on account of its satisfactory operation and sparing use of lubricants, favored increasingly for high-grade low-speed engines, is illustrated in figure 14. Drilled passages run from the middle of the main bearings diagonally through the middle of a journal and the webs of the cranks and issue in the middle of the crank pins. The connecting rod is hollow. Whenever these passages register, a part of the oil, aside from lubricating the main and the crank-pin bearings, is forced through the hollow connecting rod and lubricates the piston-pin bearing. The oil is initially forced under pressure through the covers of the main bearings. In other engines the oil is supplied through the bottom of each main bearing, permitting a simple arrangement of the oil piping and obviating the necessity of breaking pipe connections when the bearing covers are removed. The surplus oil is collected in the crank pit, flows to a filter, through a cooler, and is returned to the bearings by a positive-pressure pump, driven direct from the main engine either by gearing or side cranks. Figure 15 illustrates such an oiling system.

The lubricating system last described effectively lubricates the piston-pin bearing, which is of particular importance, as its position inside the trunk piston and in close proximity to the piston head

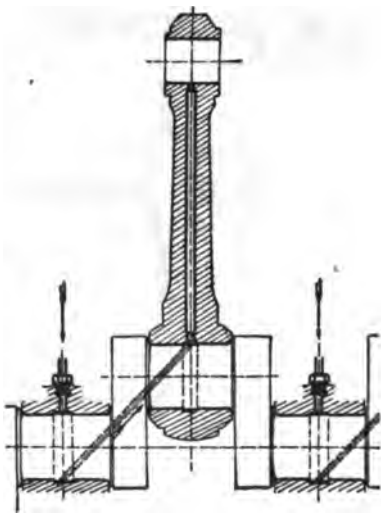


FIGURE 14.—System of lubrication especially adapted to high-speed engines.

subjects it to heat by absorption and conduction, in addition to the severe duty it has to perform regularly in transmitting the high piston pressures. Engines using centrifugal oiling rings have a separate oil supply for the piston pin, as indicated below.

In vertical engines one of the plungers of a forced-feed oil pump used for cylinder lubrication delivers oil through a feed in the

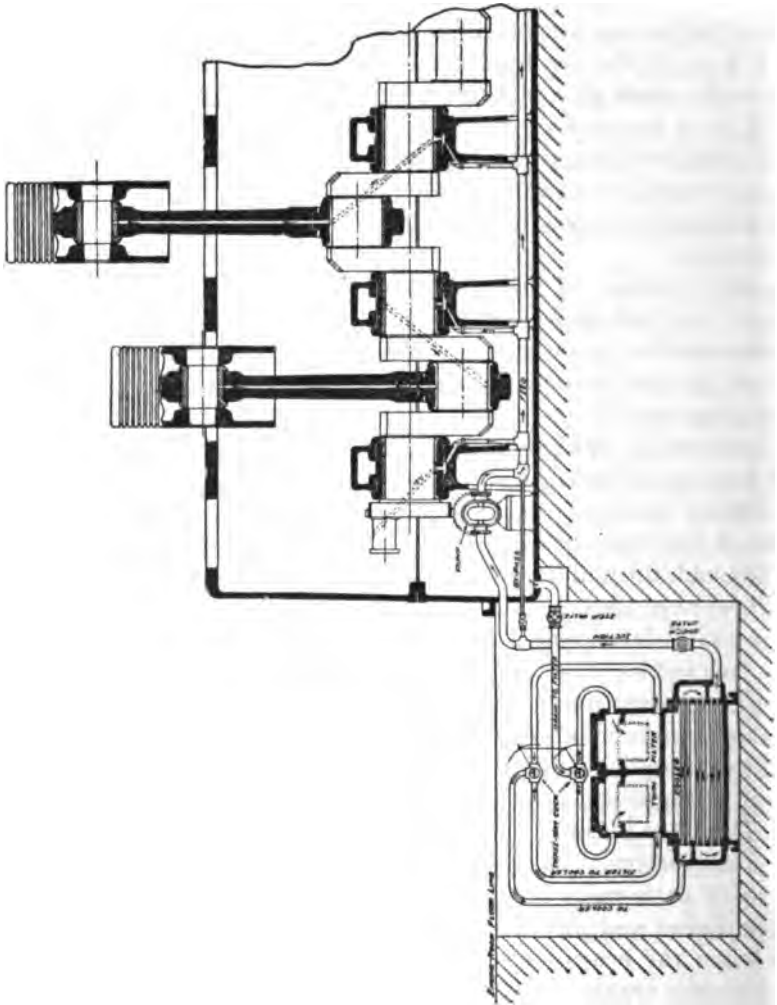


FIGURE 16.—Lubricating system in which oil is supplied through the bottom of each main bearing.

cylinder frame and the liner to a vertical groove in the piston. This groove is above the piston pin and is connected with drilled passages in the piston and the piston pin, and through these passages the oil is fed to the pin bearings.

In horizontal engines with trunk pistons the feeding of the piston-pin bearing is simple. An inclined oil-feed pipe provided externally

with a wiper is fastened to the inner wall of the trunk piston. With every stroke of the piston a measured supply of oil furnished by a separate pump plunger is carried to the piston-pin bearing.

A method of driving a rotary lubricating oil pump on a vertical engine is shown in Plate II, A.

MAIN BEARINGS.

The main bearings are usually semicylindrical, babbitted, cast-steel shells, carried by carefully machined seats cast integral with the bedplate, the shells being held down by their covers. They have no wedge adjustment, but rest rigidly in their seats. In practice it has been found difficult to keep numerous adjustable bearings in perfect alignment, and inequalities in adjustment of the different bearings have frequently led to broken shafts. With the bearings once carefully machined and aligned in the shop the wear over all bearings is uniform and light in high-grade engines, in which the bearings are flooded with oil from a forced-feed system, uniform bearing pressure being insured through an equally proportioned supply of fuel oil furnished to each cylinder. Thin brass shims are placed between the bearing shells; to take up wear a shim is removed from each side of each bearing. By raising the shaft slightly the shells can be rolled out for rebabbiting. The shop method of insuring alignment of the bearing seats is shown in Plate I, A (p. 20). The seats are chalked, and any inequality is apparent on the carefully machined arbor. Any roughness is then removed by scraping.

CONNECTING ROD.

The connecting rod is forged of soft, low-carbon steel and has a hollow center in engines using forced-feed lubrication. Hollow rods are often used, especially in small engines, to decrease weight. The crank-pin end is usually provided with a marine head, the boxes of which are either lined with babbitt or bronze, the head proper being of cast steel, and is bolted to the connecting rod. Brass shims are usually placed between the head and the rod to permit adjustment and insure the desired compression. The piston-pin end of the rod is usually slotted out of the solid to receive an adjustable box made of cast steel or cast iron lined with babbitt and bronze. Adjustment is made either by shims held by adjusting screws or by wedges. The positive rigidity obtained with square shims and adjusting screws is usually given preference over the wedge adjustment. For large engines a marine head for the piston end is sometimes preferred as offering greater ease of adjustment; however, it offers no advantages over the closed head when it has to be removed. The piston and rod of either type are pulled through the top of the cylinder

and the rod is removed by knocking out the piston pin. In horizontal engines the pistons are pulled out of the open bottom of the cylinder, so that the heads do not have to be removed, an advantageous arrangement. Some types of vertical engines are so built as to permit pulling the piston from the bottom of the cylinder.

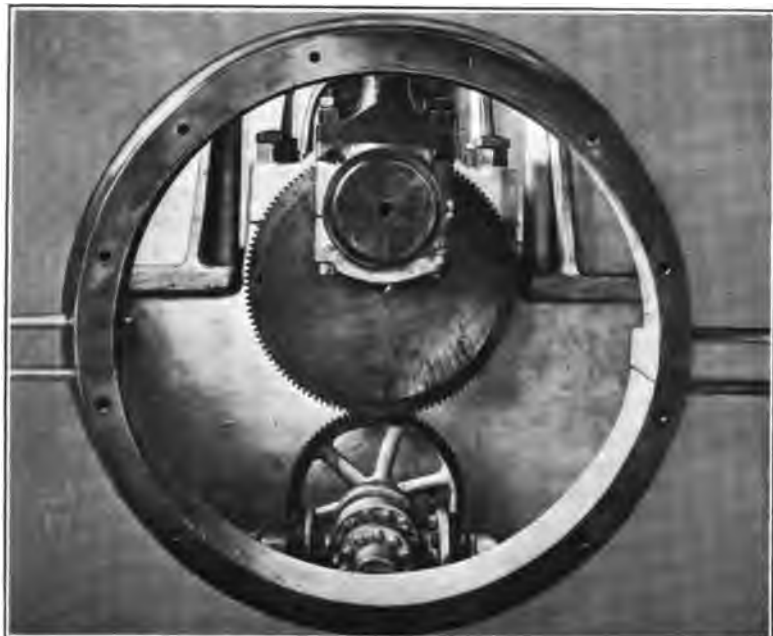
PISTONS.

On account of the relatively short connecting rod and the high pressure common to Diesel engines, the resultant side pressure on the cylinder wall is high. Therefore, the trunk pistons must be correspondingly long to keep the unit pressure within safe limits and to reduce excessive cylinder and piston wear. For this reason crossheads and guides were used during the earlier development of Diesel engines in Europe. On account of its many advantages the trunk piston has been adopted and is used in engines, even of 250 horsepower per cylinder and in units of 1,000 horsepower. The trunk piston is more easily provided with a greater bearing surface than a crosshead, thus insuring less wear; the lubrication under pressure of a cylindrical guiding surface is more effective than with open guides. The piston moves over perfectly cooled walls, whereas the crosshead tends to heat more readily and when once hot is not easily cooled; moreover, water-cooled crossheads complicate construction. The use of crossheads and guides greatly augments the height or the length of the engine, increasing its cost. For large engines this construction is used, as it affords greater accessibility and ease of adjustment, the crosshead guide in such engines being water cooled.

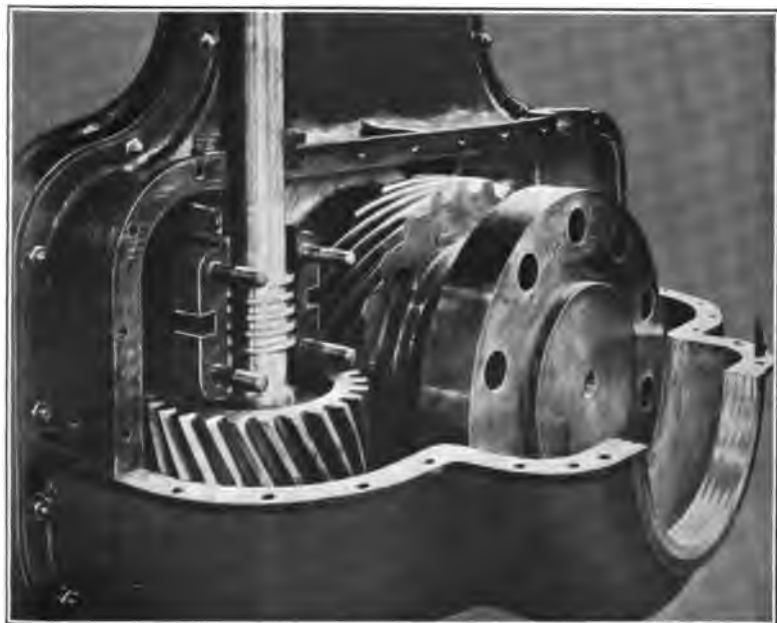
To confine the wear to an easily and cheaply replaceable part of the piston, in some constructions the piston on the side subjected to the reaction pressure is provided with soft, cast-iron wearing shoes, which can be raised slightly by the interposition of thin sheet-metal shims, when shoes and cylinder are worn. However, this construction is not extensively used, as, with good design and workmanship and thorough lubrication, the wear is slight and is noticeable only after years of work. Cylinder wear is usually due to an excessive sand or ash content in the fuel oil used and not to piston side pressure.

The cylinder liner is cast sufficiently heavy to permit of re boring.

From five to seven snap rings are used to seal the cylinder. To hold the rings closed they are pinned together at the lap joints. They are pinned to the piston by a small dowel set in a hole in the grooves. Such fastening is always necessary in two-cycle engines to prevent the ends of the snap rings from shifting around the piston and thus traveling across the exhaust ports and catching in them while they are being uncovered or covered by the piston.



4. METHOD OF DRIVING ROTARY LUBRICATING OIL PUMP IN A VERTICAL ENGINE. PUMP IS SHOWN AT BOTTOM OF LARGE CIRCULAR OPENING.



B. VERTICAL GOVERNOR SHAFT ON A VERTICAL ENGINE, DRIVEN THROUGH HELICAL GEARS FROM THE MAIN SHAFT.

At the bottom of the piston another ring is placed, serving as an oil-wiping ring and preventing the drawing of splashed oil from the crank case and the reciprocating parts into the engine cylinder. This feature is of importance in a high-speed engine with a closed crank case.

Pistons of small size are cast in one piece; those of larger engines are cast in two pieces. The limiting size for the one-piece and the two-piece construction differs widely with different builders. In the two-piece construction the piston top with the snap rings comprises about the upper one-third of the piston. In the lower piece is placed the gudgeon pin. That part of the piston subjected to intense heat and great wear by the impinging of the fuel and air jets can thus be cheaply replaced. The top of the piston is either a plane surface or has a concave depression. To reduce the clearance space to a minimum and to obtain the required compression, the piston is permitted to come close to the head when at its top center by cutting out recesses in the top of the piston for the air inlet and exhaust valves. To prevent excessive wear of the central top part of the piston, upon which the fuel and the injection air impinge, a nickel or steel plate is sometimes cast into that part. Other constructions provide a small cone to distribute the fuel radially over the entire combustion space and to improve the combustion of the fuel. The construction of the piston is largely a matter of practical experience. The interior of the trunk is provided with reinforcing ribs for strength as well as for radiating heat readily. On account of the repeated heat changes and high pressures, the piston is heavily taxed. Notwithstanding, all material must be used judiciously and in the right places, as any excess iron will serve only as a heat accumulator and subject the piston to severe internal stresses caused by differences in temperature in different parts of the piston. A light piston is also desirable to reduce the weight of the reciprocating masses.

Large pistons are jacketed and cooled by oil, water, or air and water. The oil or water is either circulated through the jacket space, or the water is sprayed with air against the inside surface of the piston top.

In some engines oil is used for piston cooling, to prevent contamination with water of the lubricating oil in the crank case, should the moving pipe connections supplying the cooling medium spring a leak. However, the use of oil as a cooling medium is not general; as its specific heat is only about half that of water. Water is also far superior for transmitting heat. As satisfactory mechanical constructions have been developed that insure a continuous supply of cooling water to the piston with small likelihood of leaks, the use of oil has been superseded by that of water even by builders who have long adhered to the former practice.

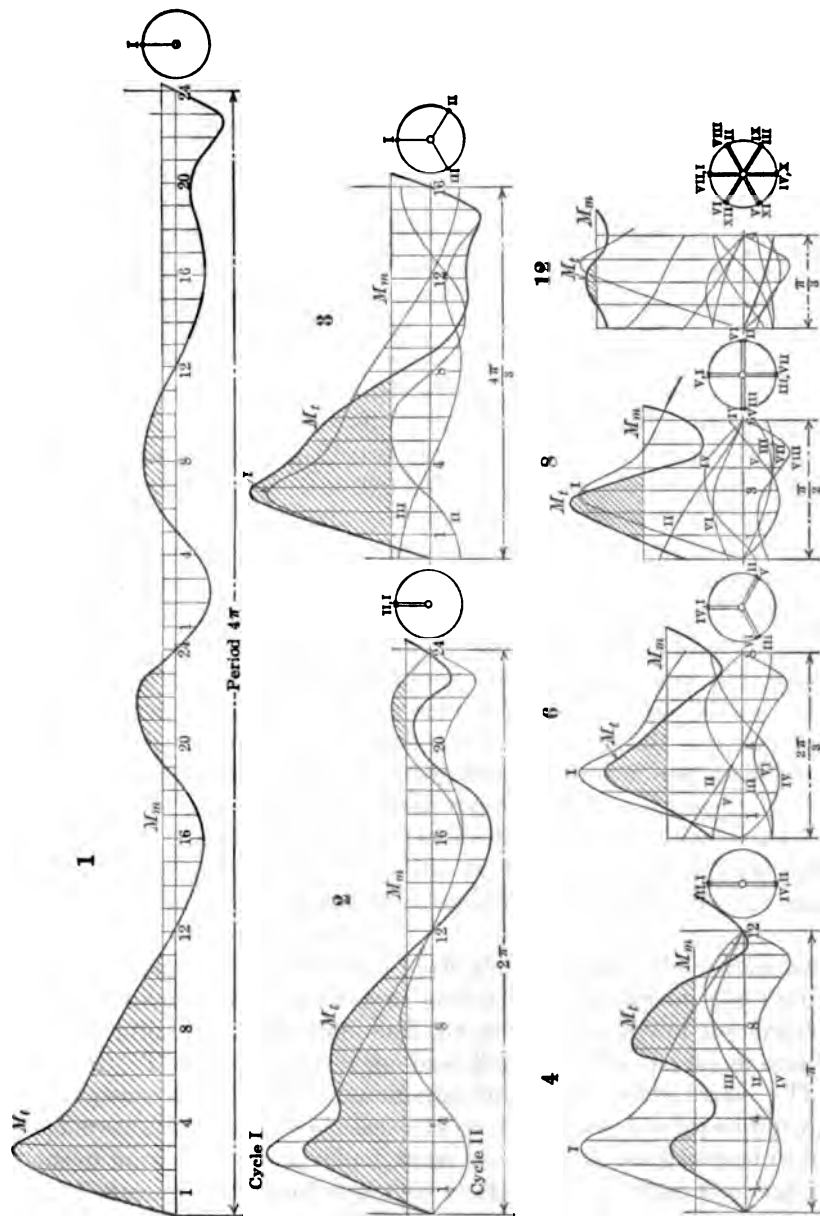
The water for piston cooling may be circulated through telescopic pipes, with the stuffing box a moving part of the piston, or the stuffing box may be attached to the frame and the water be supplied to the piston through hollow walking arms through which the water flows to the pipes leading into the piston. A pump may be actuated from the crosshead and the water be carried to the piston through the hollow piston rod. In another cooling system water is sprayed by air against the heated piston surface, not enough water being used to fill the water space of the piston. The excess water drains off through a pipe surrounding the spray pipe, no stuffing box being used.

There is no uniform practice among manufacturers regarding the size of engine at which a change from air-cooled to water-cooled pistons is made. Some use water-cooled pistons when the cylinder reaches 125 horsepower. Others build air-cooled pistons for cylinders of 250 horsepower. In construction involving a closed crank case, which gives little opportunity for air-cooling of the piston, the former practice is favored, whereas for horizontal engines in which the interior of the trunk and the part of the piston extending outside of the cylinder are exposed to the air the higher limit may be used.

Pistons of two-stroke cycle engines, in which the scavenging air is admitted through ports in the bottom of the cylinder, have special shaped tops to direct the flow of the scavenging air.

In horizontal engines, the ratio between the length of connecting rod and that of the stroke is generally greater than in vertical engines; a ratio of 3 is common, so that the reaction pressure against the side of the cylinder is usually less than in vertical engines, even allowing for the additional pressure due to the weight of the piston, which is always slight compared with the total side pressure on the cylinder.

The cylinders are lubricated by providing each cylinder with a force-feed pump operated by a reducing motion from the end of the piston, or with a multiple-plunger pump driven by an eccentric device or direct from the cam shaft, one plunger being provided for each working cylinder and one for each air-compressor cylinder. In some engines two plungers per cylinder are provided, one for each pair of the four oil feeds grouped around the cylinder. Vertical two-stroke Diesel engines usually have a separate oil feed above and below the exhaust ports to prevent any excess of lubricating oil from being swept through the exhaust ports. Horizontal two-stroke engines usually have the exhaust ports in the sides of the cylinder, the parts of the cylinder serving as top and bottom guides being solid; the lubricating oil can then be so distributed over the bearing surfaces as to occasion little if any loss through the exhaust ports and separate oil feeds are not needed.



CRANK-EFFORT DIAGRAMS FOR FOUR-STROKE ENGINES WITH 1, 2, 3, 4, 6, 8, AND 12 CYLINDERS.

FLYWHEEL.

Flywheel construction does not differ from that common to gas-engine practice. To insure a fairly uniform speed, a flywheel becomes necessary. On account of the high compression and mean effective pressures, the cranks are subjected to severe duty and transmit correspondingly high efforts. During the power impulse, excess power must be stored in the flywheel to be released during the compression stroke and during the other two unproductive strokes in engines with a four-stroke cycle. One-cylinder engines, especially if great uniformity of speed is demanded, as in driving two or more alternators in parallel, require an excessively heavy flywheel. Concentration of all the power in one cylinder also greatly increases the comparative weight of the engine.

Commercial as well as technical reasons, therefore, impose limitations on the size of single-cylinder units. These seldom exceed 150 horsepower and then are usually confined to horizontal engines for industrial uses allowing a liberal variation in speed. Two-cylinder units are built in sizes of 50 to 300 horsepower, and three and four cylinder units in sizes from 100 horsepower up. Multicylinder units, by dividing the work among a number of cylinders, have a better rhythm and greatly reduce the individual tangential forces received by the crank, and therefore require a much lighter flywheel. Thus, the flywheel of a one-cylinder engine, if used on a four-cylinder engine with the same size of cylinder and four times larger power, will cause a cyclic regularity nearly 400 per cent better than that of the one-cylinder unit. Multicylinder units are considerably lighter; the smaller-sized cylinders allow shorter strokes and therefore higher rotative speeds, and these reduce the required flywheel weight.

Plates III, IV, and V show how the cyclic regularity is affected by (a) the number of cylinders; (b) the cycle of the engine, whether two-stroke or four-stroke and whether single or double acting; and (c) the setting of the cranks.

The abscissas denote the time cycle during which the forces act, and the ordinates show the forces acting tangentially on the crank pin. The number of cylinders and the setting of each of the cranks are shown. The area between the base line and *M* is the mean crank effort or power furnished by the engine during the entire cycle. The shaded areas represent the excess work of the engine developed during periods of maximum work, which must be stored in the flywheel to be given off during periods of negative work; that is, when the engine produces insufficient power and consumes power.

In Plate III, the numbers 1, 2, 3, 4, 6, 8, and 12 denote the number of cylinders of same size in the four-stroke engines compared. The shaded areas become successively smaller with an increase in the

number of cylinders, notwithstanding the fact that the mean crank effort corresponding to the number of cylinders is proportionately greater. If all these engines were to have the same degree of regularity, the flywheels would have to be dimensioned proportionately to the shaded areas, clearly illustrating the much reduced flywheel weight required by multicylinder units. As previously mentioned, such units give much greater uniformity of motion and rhythm with a corresponding decrease in the weight of reciprocating masses and dimensions of units, thus permitting great compactness with relatively small head room. These are the factors principally responsible for the general adoption of multicylinder units and not the inability to concentrate only small power in one cylinder, as European manufacturers have developed cylinders that yield 2,000 horsepower.

The crank-effort diagrams in Plate IV are of single-acting two-stroke engines with 1, 2, 3, 4, 6, and 8 cylinders. These engines require still lighter flywheels, mainly because they receive a power impulse per cylinder during each revolution, instead of every other revolution as in engines having a four-stroke cycle, represented in Plate III.

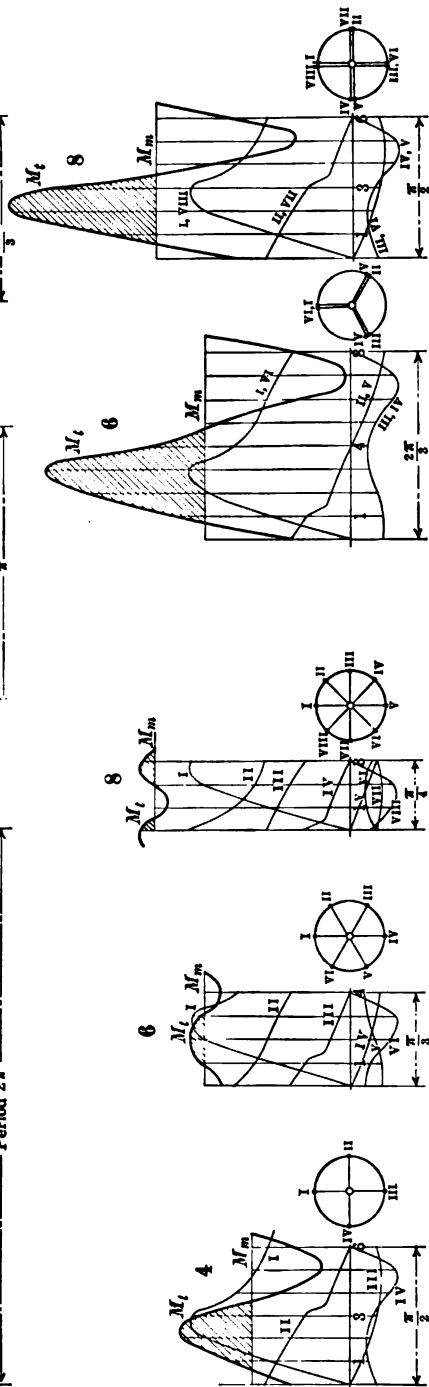
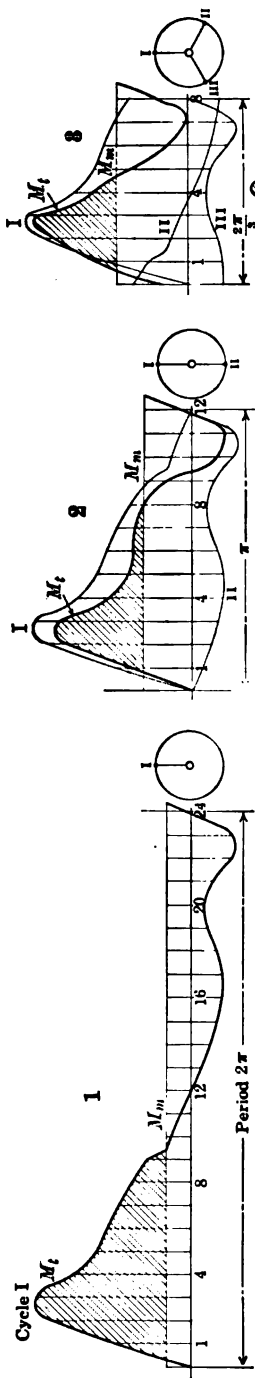
Although in all the diagrams of Plate III and in the first six of those in Plate IV the cranks are shown set so that the power impulses are equal time intervals apart, the last two diagrams in Plate IV representing six and eight cylinder engines illustrate a crank position in which two power impulses are transmitted to the shaft simultaneously. This setting increases correspondingly the excess work to be stored in the flywheel and requires a proportionately heavier flywheel.

In Plate V are shown diagrams of double-acting, two-stroke engines with 1, 2, 3, 4, and 6 cylinders; two six-cylinder engines are shown, one receiving only one working impulse at a time per cycle, the other receiving two simultaneous working impulses per cycle. These diagrams show that double-acting, two-stroke engines require the smallest flywheel weights.

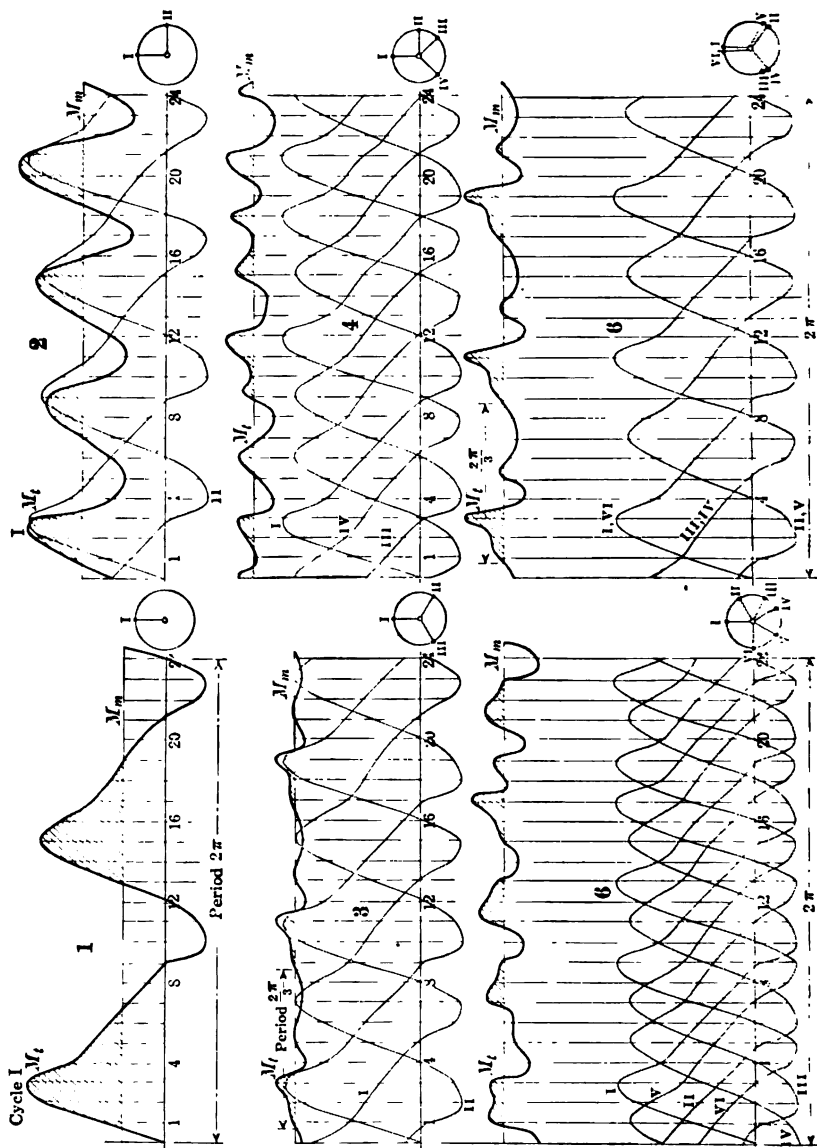
A one-cylinder, single-acting, two-stroke engine generally requires about the same flywheel effect as a two-cylinder four-stroke engine; and a given number of cylinders in a double-acting four-stroke engine imparts the same regularity to it as does an equal number of cylinders in a single-acting two-stroke engine.

The irregularity factor may be defined as the ratio of the difference between the maximum and the minimum velocities to the average velocity of the flywheel during one revolution and is expressed by the formula $\frac{V_{\text{maximum}} - V_{\text{minimum}}}{V_{\text{mean}}}$; it therefore expresses the maximum variation in velocity during one revolution and is also known as the pulsation of the prime mover.

Cycle I



CRANK-EFFORT DIAGRAMS FOR SINGLE-ACTING TWO-STROKE ENGINES WITH 1, 2, 3, 4, 6, AND 8 CYLINDERS.



CRANK-EFFORT DIAGRAMS FOR DOUBLE-ACTING TWO-STROKE ENGINES WITH 1, 2, 3, 4, AND 6 CYLINDERS.

The maximum variation from uniform speed differs widely for different classes of work required of the engine. The irregularity factor may vary as follows:

For driving pumps, punches, shears, and machines requiring a low degree of regularity, one-twentieth to one-thirtieth.

Machine tools, looms, and paper machinery, one thirty-fifth to one-fortieth.

Spinning machinery, according to class of work, one-sixtieth to one one-hundredth.

Direct-current generators, one one-hundred-and-fiftieth.

Alternating-current generators, one two-hundred-and-fiftieth to one three-hundredth.

Making allowance for the different classes of power service that Diesel engines have to perform, manufacturers usually provide standard light, medium, and heavy flywheels for all engines, the medium and heavy flywheels of smaller engines being also available as light and medium flywheels for progressively larger machines. The set of flywheels used on one-cylinder engines is also used on multicylinder units with cylinders of same size.

One-cylinder units should not be used if the permissible variation has to be less than one-fiftieth; two-cylinder units may be used if the factor of irregularity does not have to be less than one one-hundredth; one two-hundredth is the limit for three-cylinder units, and one three-hundredth or less for four-cylinder units. Naturally, three and four cylinder units can perform the work of one or two cylinder units with a much lighter flywheel. The foregoing figures apply also to engines having a two-stroke cycle but with half the number of cylinders.

Limitations as to weight and dimensions of engine and flywheel, floor-space requirements, the more perfect balancing of the much-reduced reciprocating masses, and the great compactness will often decidedly favor the multicylinder unit, especially when the Diesel engine is to be directly connected to an alternating-current generator and two or more units are to operate in parallel. Multicylinder units also permit a high rotative speed, which lowers the flywheel weight and increases the permissible variation of the engine through the reduction of the number of poles in the generator.

CYLINDER HEAD.

In engines having a four-stroke cycle the usual practice is to place all valves in the cylinder head; in engines having a two-stroke cycle the exhaust valves, and in some engines the scavenging valves also, are placed in the cylinder and the other valves in the head.

The cylinder head of vertical engines (fig. 16) is a cylindrical water-jacketed body, with flat bottom and top, and is secured to the cylinder frame with liberally proportioned stud bolts. Openings for the different valve cages in the top of the head are grouped symmetrically whenever feasible; the fuel-injection valve is usually in the center,

and the air and exhaust valves on each side, in the same cross-sectional plane and in line with the axis of the engine. The opening for the starting valve is usually placed in front of the central opening, in the same plane, and at right angles to the plane through the other three valve openings. When the available head area is too small to permit the fuel-injection, air, and exhaust valves to be placed in the same plane, the fuel valve is placed slightly forward, permitting the other valves to be placed closer together, so as to allow the required area.

A flat, smooth, bottom surface in a head results in a simple combustion chamber not split up with ribs and pockets. Such a surface is a great aid in reducing the clearance space to the required minimum and affects favorably the combustion of the fuel. Such heads can

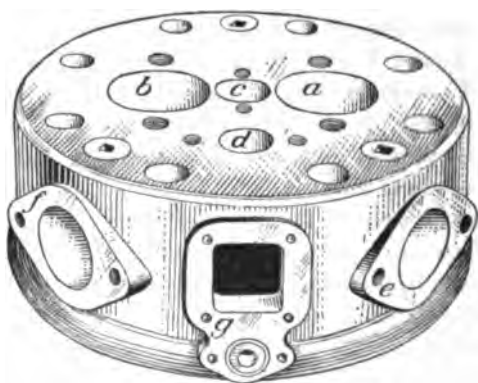


FIGURE 16.—Cylinder head of vertical Diesel engine. *a*, opening for cage for air-admission valve; *b*, opening for cage for exhaust valve; *c*, opening for fuel valve; *d*, opening for starting valve; *e*, facing for air-pipe connection; *f*, facing for exhaust-pipe connection; *g*, facing for water-pipe connection and handhole.

be easily designed for vertical engines, permitting the use of vertical valve stems, which insure positive and easy seating.

The advantages of a flat head are considered sufficient by some manufacturers of horizontal engines to outweigh the advantages of vertically seating valves. Others attach more importance to vertical valves and follow gas-engine practice in the construction and operation of the valves and the cylinder head. The air

valve is placed in the top of the head and the exhaust valve in the bottom, the fuel valve being in front, in the center.

The starting valve is then placed either in the front of the head or on the side halfway between the air and the exhaust valves. The use of a gas-engine head provides a larger water space; however, this is much constricted by the numerous walls surrounding the valve openings if the valves are all placed in the top or end of the head. Efficient cooling of the head is then rendered more difficult. Hence, it is advisable, if a supply of pure water is not available, to use distilled water for cooling the cylinder head and the cylinder, and to recool the water for reuse; otherwise trouble may be experienced from precipitation of lime and magnesia salts allowing the head to become overheated and causing it to crack.

The placing, form, and direction given to the air and the gas passages differ in accordance with the space available and the design adopted for the air-intake and the gas-exhaust pipes.

The heads of two-stroke engines have generally the same cylindrical form as those of four-stroke engines, but in the former there are no openings for the exhaust valves, which are placed in the lower part of the cylinder.

The scavenging valves are placed in the head, and as the volume of air required for a given cylinder volume is 1.2 to 1.4 times greater

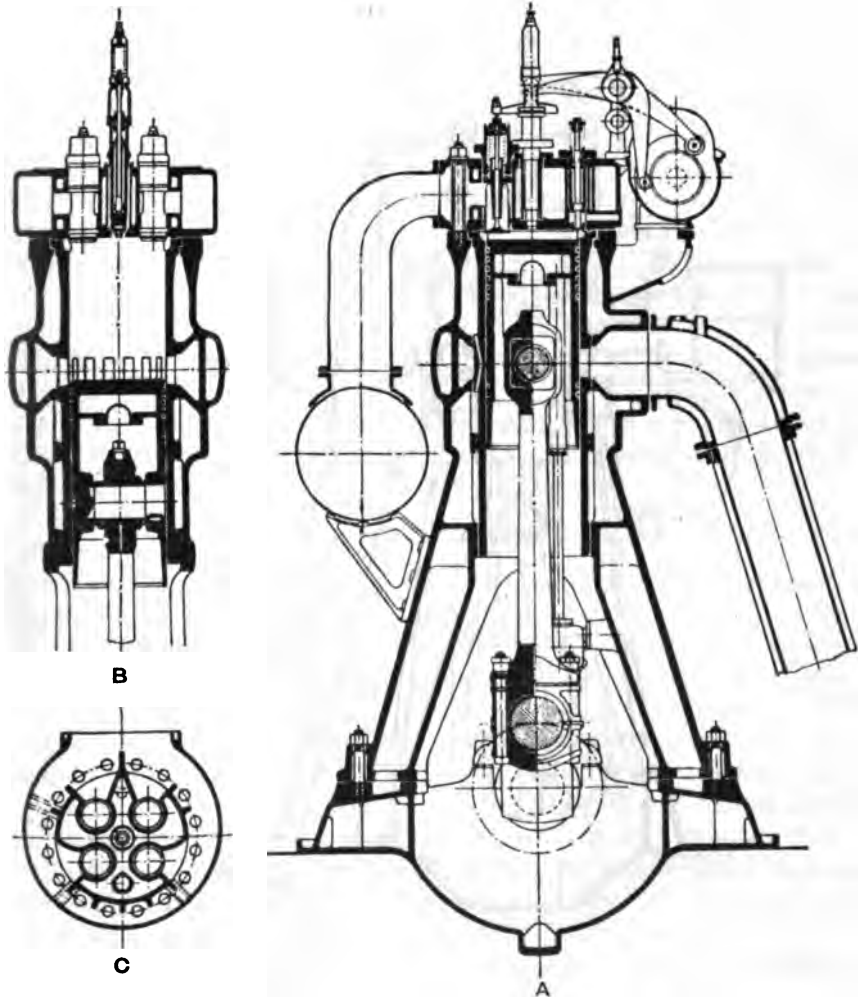


FIGURE 17.—Two-cycle engine with air admission through cylinder head, showing scavenging air valves. Exhaust ports uncovered by piston are shown at A.

than for engines having a four-stroke cycle, and as the time available for the scavenging and the refilling of the cylinder with fresh air is short, large valves are needed. The area of the head sets definite limits for the size and placing of these valves. To facilitate their simultaneous opening and simplify the valve gearing, the valves are grouped in pairs, and four valves are symmetrically

distributed over the top of the head (fig. 17). The fuel and the starting valves are placed as in engines having a four-stroke cycle. The placing of so many valves in the cylinder head greatly restricts the water space and renders cooling more difficult; the valve area is also limited by the head area, and high air velocities and correspondingly high scavenging air pressures are necessary. These

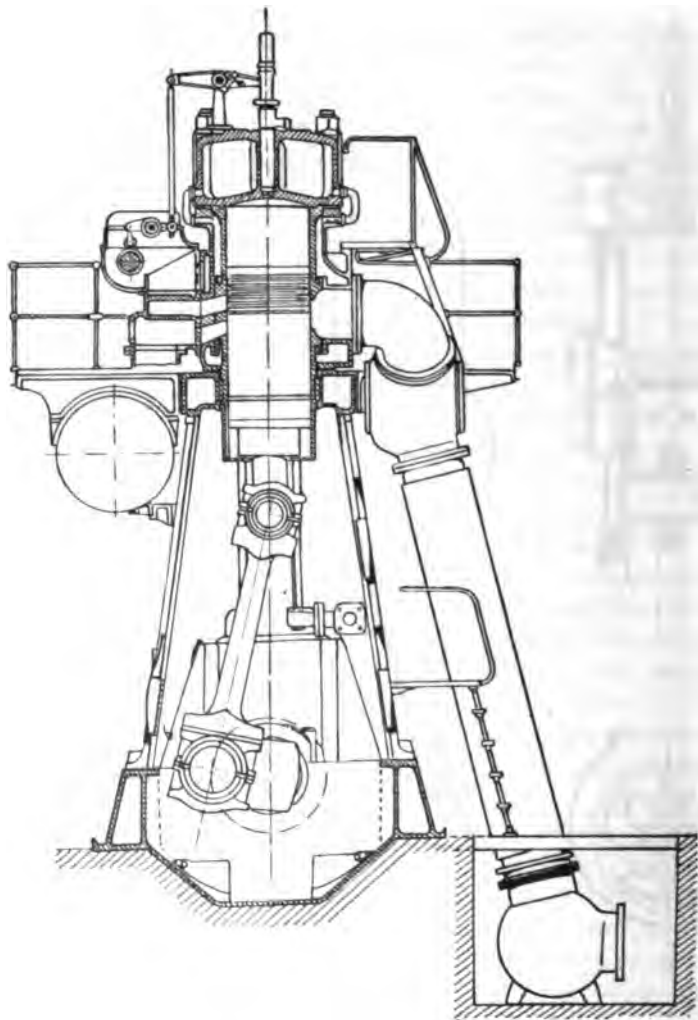


FIGURE 18.—Two-stroke engine with two tiers of air valves. Air valves opposite exhaust ports.

disadvantages are partly offset by the effective scavenging obtained with valves opening into the top of the cylinder. A more recent engine construction simplifies the cylinder head by placing the exhaust ports as well as the ports for the scavenging air in the middle of the cylinder (fig. 18). Two tiers of scavenging ports are provided, of which the lower one is uncovered by the piston, whereas

the upper is controlled by a valve actuated by the cam shaft. The lower tier of ports is opposite the exhaust ports, and in direct communication with the air receiver when uncovered by the piston. After the piston has traveled past the exhaust ports and the lower row of scavenging ports and closed these, the upper row of air ports is still uncovered, and air can be forced into the cylinder, increasing the specific output of the engine. The valves controlling the upper tier of ports are in a region of low pressure, and therefore low temperature. During the periods of compression and high temperature these valves are covered by the piston, whereas with the scavenging valves in the head there is a certain amount of danger of leaking valves letting fuel vapors or gases enter the scavenging air receiver and cause explosions. The valves closing the upper tier of air ports prevent the entrance of exhaust gases into the scavenging air receiver before the pressure of the exhaust gases is reduced to atmospheric pressure with the uncovering of the exhaust ports by the piston. The absence of scavenging valves in the cylinder head simplifies its form greatly, as all surfaces can be equally and thoroughly cooled; and with the elimination of internal ribs between the valve seats, stresses due to unequal expansion are avoided. Only the fuel-injection, starting, and relief (safety) valves are in the cylinder head.

AIR INLET AND EXHAUST VALVES.

Air inlet and exhaust valves (fig. 19) are usually housed in flanged cages bolted to the cylinder head. A cage contains the valve seat and the valve-stem guide. The upper part of the barrel is machined and serves to guide the carefully centered spring cap. The latter is screwed to the valve stem and holds the spring in place. The valves are generally made of cast iron, the stem being of steel. It is usual practice to provide valves, except those of the smallest sizes, with removable valve seats. Air and exhaust valves seldom differ, except that for engines with more than 30 or 35 horsepower per cylinder the valve-stem guide is water-cooled. Four-stroke engines are seldom built in sizes requiring the water-cooling of the exhaust valves, and two-stroke engines exhaust through ports uncovered by the piston. Figure 20 shows the customary valve construction. The spring cap has a concave depression, into which the jointed ball of the actuating lever fits.

In engines fitted with gas-engine heads, the usual practice is to house only the air valve (placed in the top of the head) in a cage. After the air valve with cage has been removed, the exhaust-valve stem, disk, and seat are accessible through the opening in the air-valve cage and can be removed through it. In two-cycle engines with gas-engine head, only the scavenging valves are in the head, and

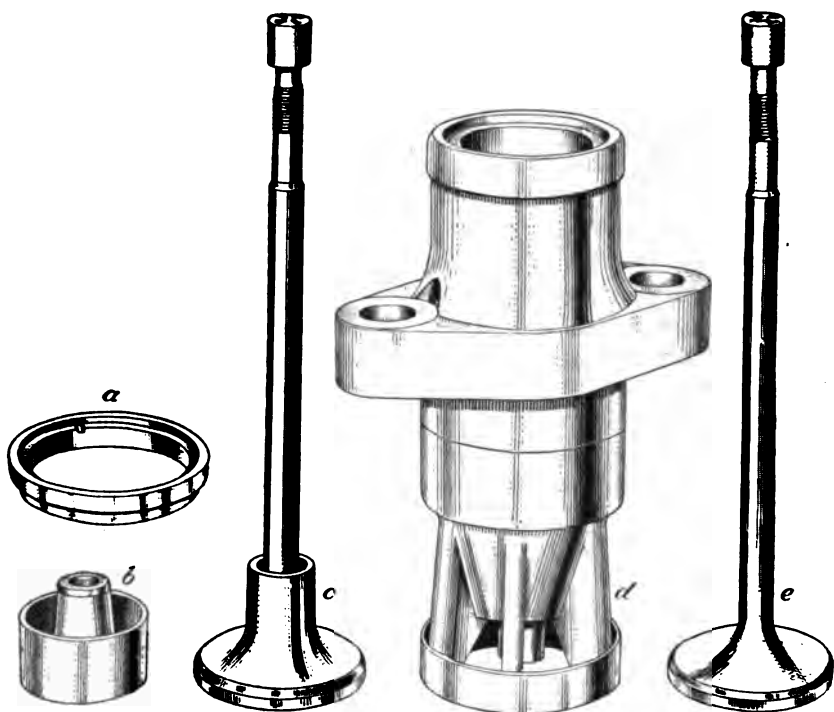


FIGURE 19.—Valves and valve fittings. *a*, removable valve seat; *b*, spring cap; *c*, exhaust valve in stem; *d*, valve cage; *e*, air admission valve in stem.

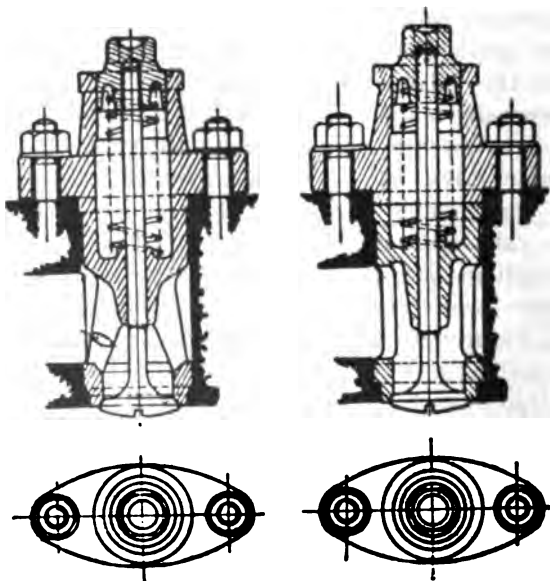


FIGURE 20.—Customary valve construction of Diesel engines.

are on its top and bottom, one directly above the other. In this type the valves are similar and are housed in cages. As only fresh air is admitted through them, and they are not fouled by the products of combustion, they seldom need to be removed. To retain a perfect seat, each valve disk is occasionally ground in its seat. The disk is provided with two holes for the insertion of a special key, or the valve stem is slotted so that a bar can be inserted, for turning the valve disk back and forth; a small amount of emery dust and oil is used in grinding.

The starting valve admits air under high pressure (500 to 600 pounds per square inch) into the engine cylinder whenever the engine is started. Hence the valve disk must seat perfectly to prevent air leakage when the valve is closed. It is usually constructed as a balanced valve; a part of the valve stem is enlarged and provided with packing rings, the air admitted pressing equally on both sides of this piston. A strong spring insures the tight seating of the valve.

Some engines are equipped to start with air at a pressure of about 250 pounds per square inch. The starting valve is fitted with a plunger that closes the valve as soon as the cylinder pressure exceeds that of the air used for starting.

FUEL VALVES.

TWO FUNCTIONS OF FUEL VALVES.

Next to the fuel pump, the fuel-injection valve is the most important and characteristic part of a Diesel engine, and on its correct construction greatly depends the satisfactory operation of the engine. It performs two distinct functions. First, it has to admit the liquid fuel into the engine cylinder at the proper time, and must therefore be an accurately timed valve. Second, it has to introduce the liquid fuel into the cylinder in a completely atomized form, and must therefore be an efficient atomizer.

The first requirement is fulfilled by storing around the tip of the fuel needle a drop of oil, which is forced ahead of the injection air into the heated atmosphere of the engine cylinder at regularly recurring intervals, and thus initiates the ignition. The second requirement is accomplished by dividing the fuel supply into numerous small fuel particles in the fuel valve, preceding atomization and injection.

ATOMIZATION OF FUEL.

As has already been explained, highly compressed air is used for atomizing the fuel oil; incidentally it serves the purpose of injecting the fuel oil into the cylinder during the process of atomization. In the atomizing nozzle of the fuel valve a part of the pressure of the

injected air is changed into velocity, which reaches its maximum at the orifice of the nozzle. Owing to the expansion of the air at the orifice there is a decided drop in temperature at this point, where the ignition of the fuel is to be initiated. This localized lowering of the temperature at the nozzle orifice is accentuated during periods of low load, when the fuel supply is diminished proportionately to the engine load, whereas the injection air supply remains constant; that is, the same as at full load, because the fuel-valve needle stays open during 10 to 15 per cent of the stroke of the engine, irrespective of the load. The relation of fuel supply to air supply is therefore changed with a change in load. It is consequently more difficult to initiate the ignition at partial than at normal engine loads. A correctly designed fuel valve must be so constructed that unailing fuel ignition at the proper instant will take place for every working impulse. Moreover, the atomization and injection of the fuel must be gradual during the entire period the needle valve is open.

Preheating of the injection air to counteract a cooling effect during fuel injection not only constitutes an element of danger owing to possible fuel ignition in the fuel valve, but results in an uneven, knocking operation of the engine as a result of explosion-like ignition of the fuel.

COMMON TYPE OF ATOMIZER.

The elements of a common type of atomizer are shown in figure 21. It comprises a fuel needle *a*, guided for a great part of its length in a cylindrical barrel *b*. The needle and the guide are carried in a flanged cage, which is slipped into the central passage in the cylinder head and is bolted to the latter. The outer diameter of the needle guide is smaller than the internal diameter of the cylindrical valve cage. The annular space *c* thus formed around the needle guide is connected with the injection air receiver and is at all times filled with compressed air. The needle guide terminates and seats against a corrugated cone *d*, provided with about 20 slots, each one-sixteenth inch wide. The cone fits the cone-shaped end of the valve cage. Above the cone are a number of perforated plate rings, *e*, so placed that the perforations of each succeeding ring are staggered with regard to those in the ring above. Each ring is provided with 18 to 20 apertures five sixty-fourths to three thirty-seconds inch in diameter. An atomizer tip is screwed over the end of the valve.

The needle valve is held against its seat by a powerful spring, provided with a suitable spring cap, a bonnet, and exterior adjusting nuts. In the body of the valve an oil passage is drilled to deliver the fuel on top of the perforated rings. In other types, to avoid the cost of drilling these long, small passages, two concentric spaces are used—the inner for air, the outer for fuel—formed by inserting a cylinder over the needle guide, thus dividing the annular space around the guide

into two compartments. Holes in the bottom of this cylinder permit the fuel to enter the inner air space and to be distributed over the perforated plates. The measured quantity of fuel oil proportioned to the engine load is deposited by the fuel pump in the fuel-valve chamber before the valve needle is opened.

The fuel oil is broken into innumerable small globules and runs from plate to plate and down into the grooves of the cone, and is dis-

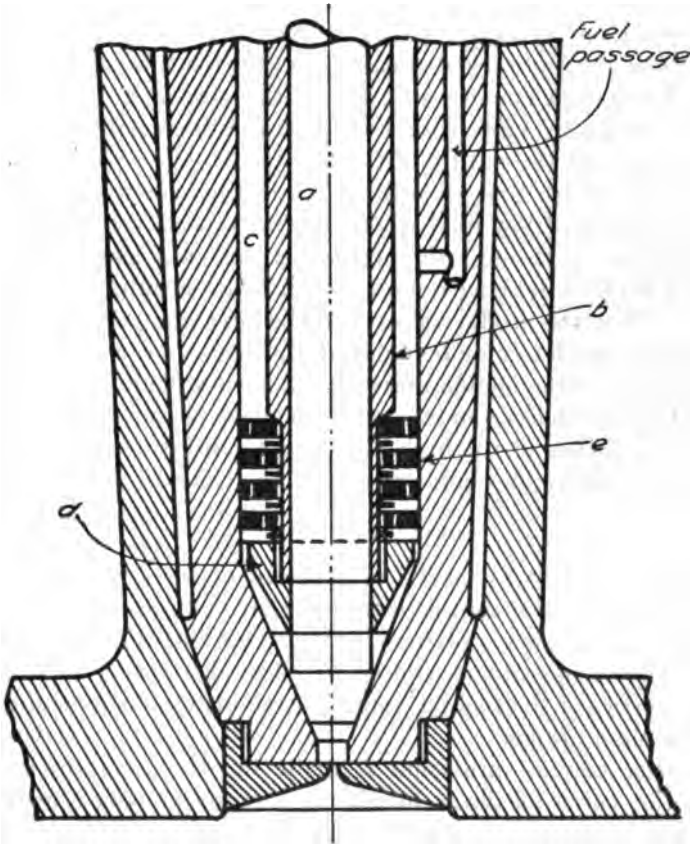


FIGURE 21.—Common type of atomizer. *a*, fuel needle; *b*, barrel; *c*, annular space; *d*, corrugated cone; *e*, perforated plate rings.

tributed over and covers the surfaces of the plates and grooves, and a small quantity collects around the root of the fuel-needle point. When the needle valve is opened, the injection air sweeps this part of the fuel before it, propagating the ignition. The air attains its greatest velocity when issuing through the nozzle. The sluggish, viscous oil particles, being in a quiescent state, resist this sudden acceleration with their inherent inertia and move relatively much more slowly than the air. The rapidly moving air currents tear these

particles to pieces, and when the air currents have attained their highest velocity at the atomizer orifice they nebulize the fuel. A small proportion of the fuel still remains behind and flows down to the top of the seat of the fuel-valve needle to initiate the ignition of the next fuel charge.

ADJUSTMENT OF PRESSURE OF INJECTION AIR.

Fuel valves of this type require a change of pressure of the injection air with a variation in load; that is, the pressure must lessen as the load of the engine is decreased if perfect combustion with colorless exhaust and smooth running of the engine is to be obtained. The pressure may vary from 40 to 70 atmospheres, depending on the engine load and the nature of the fuel.

If the pressure and quantity of injection air were not reduced during periods of low load when the fuel quantity is much less, the air would sweep through the perforated rings and the cone and clean them of all fuel, leaving none to accumulate around the fuel needle point. When the needle was opened the next time, the ignition might either fail completely or be delayed and cause a series of rapid, dull explosions. If the engine misses ignition, it will slow down, and the fuel pump will deliver an increased quantity of fuel oil, which will be ignited regardless of the high injection air pressure, but such speed variations and engine operation should be avoided.

ADJUSTMENT BY HAND.

A common way to vary the pressure of the injection air is by throttling down the air taken into the air compressor and adjusting it to the varying loads of the engine. This throttling must be done by the engine operator. The air pressure also has to be adjusted to the properties of the fuel oil, some oils requiring a higher injection air pressure than others.

In other engines provision is made for changing the height to which the valve needle is lifted. A lever is actuated to lift or to lower the fuel needle, and this lever is fulcrumed on a pin the journals of which are eccentric to the lever bearings, so that by rotating this pin to various positions, the length of opening of the fuel needle may be adjusted to the engine load. A small hand lever is keyed to one end of the pin and is provided with a "click" to hold the lever as set.

AUTOMATIC ADJUSTING DEVICES.

Such hand adjustments of the injection air pressure present no particular difficulty in small engines, but in large engines, especially those working with rapidly fluctuating loads, it is obviously impossible for the operator to adjust the air pressure quickly enough to meet the constantly changing requirements of the engine. The work of the air

compressor consumes 10 per cent and more of the power output of the engine. In large engines it is therefore desirable to proportion the volume of injection air to the engine load. This is accomplished by a patented design in which the engine governor influences not only the volume of the fuel supply, but also the volume of the injection air; the governor likewise affects the length of time during which the fuel valve is open, the period varying in proportion to the load on the engine. The governing devices are illustrated in figure 22. In the figure, *a* is a throttling valve actuated by the engine governor *b*, proportioning the amount of air taken into the scavenging pump *c* to the engine load, a part of the air from which is compressed further in the compressor *d*, which furnishes the injection air. The pressure and the quantity of the injection air are thus controlled by the engine governor. Similarly, the varying air pressure influences the travel of the piston in the servo-motor *e*, which, acting through auxiliary levers and rollers, varies their position relative to the cam. The travel of the lever actuating the needle valve is thereby varied and the length of time for which the needle valve is open is controlled.

A similar arrangement can be used in engines having a four-stroke cycle, the volume of air going to the injection air compressor being throttled by the governor, and the servo-motor being operated with air from the first stage of the compressor.

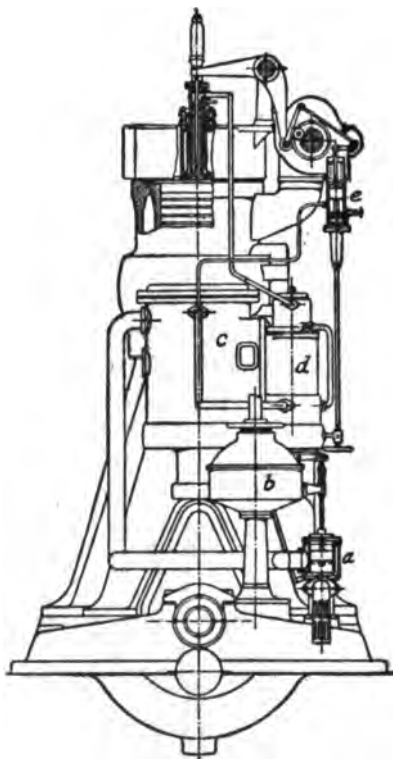


FIGURE 22.—Patented throttling devices for Diesel engines. *a*, throttling valve; *b*, engine governor; *c*, scavenging pump; *d*, compressor; *e*, servo-motor.

METHODS OF FUEL INJECTION.

HESSELMANN'S FUEL INJECTOR.

To insure an unfailing correctly timed ignition of the fuel charge, Hesselmann, of the Swedish Diesel Engine Co. (Aktiebolaget Diesel's Motorer), developed the fuel injector shown in figure 23. In this valve the fuel is admitted close to the end of the fuel needle. For this purpose the valve has two annular air spaces, connected with air passages at the top. The fuel is deposited in the bottom of the

outer space and under the influence of the full injection air pressure is forced through a number of small passages, from the orifices of which it is swept off by the passing air currents in the inner space, as soon as the valve needle is opened. The fuel is in this manner thoroughly nebulized, and a small quantity left behind accumulates around the root of the needle point to initiate the ignition of the succeeding fuel charge with the opening of the needle.

NEEDLE VALVES WITH ANNULAR FUEL PASSAGES.

In other forms of fuel valves, the same object is obtained by the use of annular fuel passages around the conical needle-valve seat, the

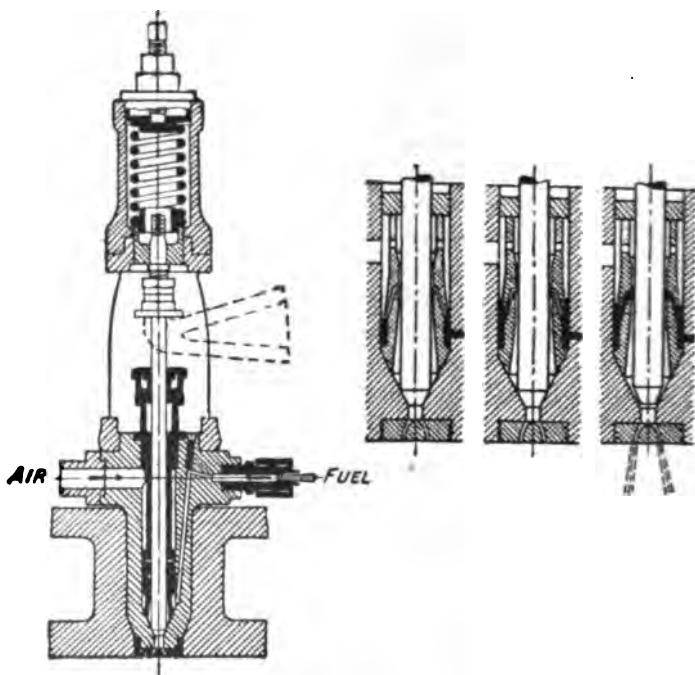


FIGURE 23.—Fuel injector developed by Hesselmann.

oil in which is under the influence of the injection air pressure. When the fuel valve is opened, part of the pressure is changed into velocity; the resulting differential in pressure between the injection air that presses on the fuel and that which travels through the nozzle forces the fuel out of the annular passages through radial openings into the air stream, by which it is picked up and atomized. A small quantity of oil left around the needle point on the closing of the valve is used for igniting the next charge.

Fuel valves of this type permit considerable load fluctuations without necessitating appreciable change in the injection air pressure. They do not, of course, effect a reduction in the quantity of air used for injecting the fuel, which would be practically constant at all loads

unless controlled by one of the methods previously described. In large engines, the saving of power effected by supplying only the required volume of injection air is important, although it necessitates adjustment to produce the most perfect ignition and combustion of the fuel. This consideration led to the automatic mechanical control of the injection air supply for large engines (fig. 22).

In the use of the fuel valves described, the fuel has to be pumped into the fuel chamber against the injection air pressure, which may vary from 40 to 70 atmospheres, the higher pressure corresponding to full load, so that the fuel pump must always work at a pressure appreciably higher than the injection air pressure. Such valves are also known as closed-nozzle valves, because the nozzle must be kept closed with the needle, except during the extremely short period of fuel atomization.

USE OF OPEN-NOZZLE FUEL VALVE.

Another type of fuel valve, known as the open-nozzle type (also called the Lietzenmayer system, after its designer), uses an open fuel nozzle, as its name indicates. In fuel valves of this construction the fuel does not work against the high injection air pressure, but is subject merely to the small pipe resistance and low lift necessary to deposit the measured amount of fuel in a recess of the fuel-valve chamber in front of the open nozzle; this is usually done during the suction stroke of the engine, when there is the least resistance (pressure) in the valve chamber. When the time for the injection of the fuel arrives, a valve, similar to the needle valve above described, is opened, admitting the injection air into the fuel chamber; the air, in passing over previously deposited fuel, impinges on it, picks it up, and, forcing it into the atomizer, nebulizes it. A valve of this type is shown in figure 24.

EQUIPMENT OR METHODS PERMITTING USE OF COAL TAR AND COAL-TAR OILS.

All of the valves mentioned work well with fuels that are easily ignited. Certain liquid fuels, such as coal tars and coal-tar oils, are ignited with difficulty, and when the engine is cold (before it is started) or at low load ignition fails entirely. The size of the engine cylinder, as well as the speed of the engine, have considerable influence on the success of the ignition. Small cylinders radiate the heat relatively faster than large cylinders, the ratio of the area of cylinder walls and head to the volume of the combustion space being greater than in large engines. Consequently the combustion space of small engines is cooled faster than that of large engines, especially during periods of low load. High engine speed shortens the time available for ignition and burning of the fuel, which in some engines must be done in less than one one-hundredth of a second.

STARTING ENGINE WITH GAS OIL.

To permit the burning in Diesel engines of coal-tar oils (a by-product of coking and gas-works operations and a distillate of coal tar), two methods have been used. One is to start the engine with gas oil and after it has become hot change to tar oil. The governor

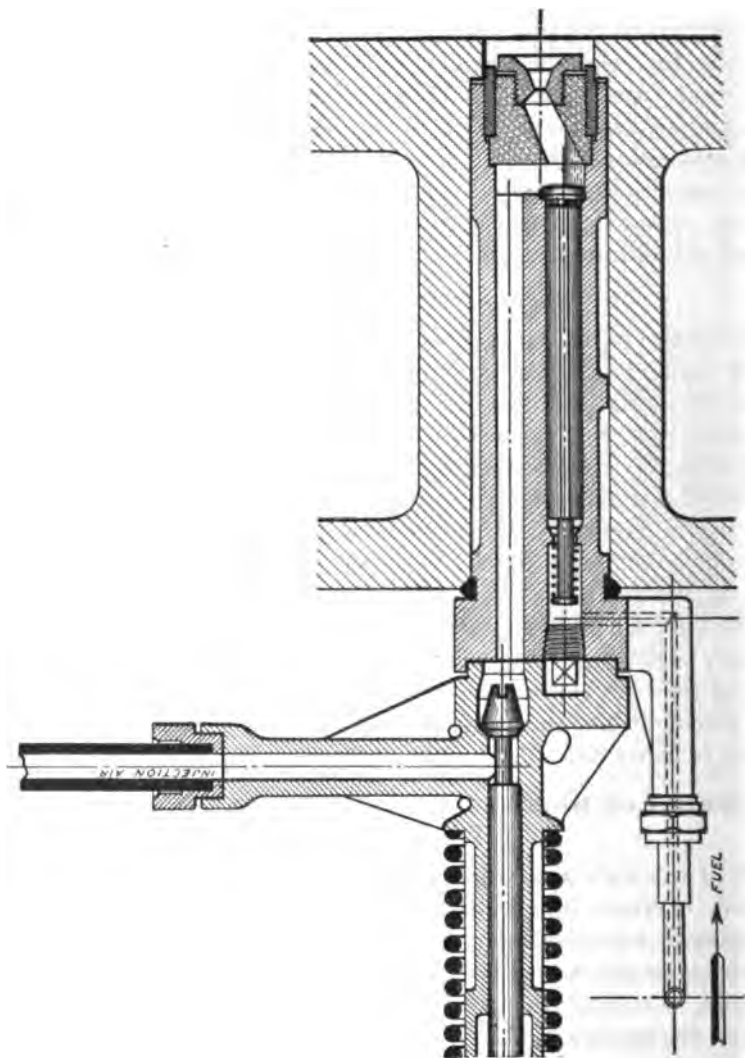


FIGURE 24.—Open-nozzle fuel valve.

and the fuel pump are so designed that with a lowering of the load the fuel is changed to gas oil and vice versa. This method has the disadvantage of causing an unnecessarily large quantity of gas oil to be consumed if the engine works a large part of the time at fractional loads.

USE OF SEPARATE FUEL PUMPS.

In another system separate fuel pumps are used for the gas oil (also referred to as ignition oil, because it initiates the ignition) and the coal tar or tar oil. The two oils are stored separately in the fuel valve, the ignition oil being so placed around the edge of the end of the valve needle that it will always enter the combustion space first, and thus initiate the ignition. The fuel is injected with compressed air as in the other valves previously described. The proportion of gas oil to tar oil used varies with the load. At no load the gas oil amounts to as much as 20 per cent of the tar oil; at quarter load it drops to 15 per cent, at half load to less than 10 per cent, at three-quarters load to 4 per cent, and at full load and at overload to about 3 per cent. During average operating conditions the consumption of gas oil seldom exceeds 10 per cent, and is usually 7 per cent. The combined fuel consumption, on a basis of heating value, is nearly the same as if only gas oil were used. Both pumps are, of course, under the influence of the engine governor. The nicety and sensitiveness of this highly successful process may be realized from the fact that the volume of gas oil injected ahead of the tar oil is exceedingly small. Thus for a 50-horsepower cylinder the total volume of fuel oil injected for one power stroke at full load is only 2 c. c., or a cube of the size shown in figure 25. The part of the cube corresponding to the volume of gas oil used during usual load conditions (80 fuel charges producing 160 revolutions per minute) represents only 0.06 to 0.2 c. c.

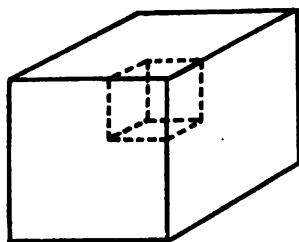


FIGURE 25.—Proportionate volumes of tar oil and of gas oil used in one power stroke of 50-horsepower cylinder at full load. Large cube represents volume of tar oil; small cube represents volume of ignition (gas) oil.

If the fuel oils, tar, or tar oils are highly viscous and do not flow readily to the fuel pump and to the fuel valve and resist atomization, gas oil is used to start the engine and run it until hot, and the hot discharge water from the engine jacket is used to heat the viscous oil to increase its fluidity. Then the gas oil is displaced by the viscous oil. This system requires a separate small tank for the gas oil, in addition to the one used for the viscous fuel oil, and the pipe system must be so arranged that either fuel can be supplied to the fuel pump. When the engine is to be shut down, the gas oil should be switched on and the engine operated on gas oil until all of the viscous oil has been displaced. This arrangement works satisfactorily except when the operator forgets to change over to gas oil before stopping the engine or in the event of an unforeseen sudden shutdown owing to some engine

trouble. These contingencies are so rare that separate fuel pumps and pipe systems for the two kinds of fuel are seldom provided. The method mentioned should be used on all fuel oils the viscosity of which at ordinary temperature ($20^{\circ}\text{C}.$) exceeds 3° Engler.

RETARDING INJECTION AIR AND FUEL.

In the cylinders of large engines the fuel needle can be so governed that the entrance of the injection air and fuel into the combustion space is retarded and gradual, preventing excessive use of injection air cooling of the atmosphere at the point where the ignition must be maintained; by these means tar oil can be ignited without the use of ignition oil.

The methods of using tar oils described are of no immediate importance in the United States, where an abundant supply of petroleum

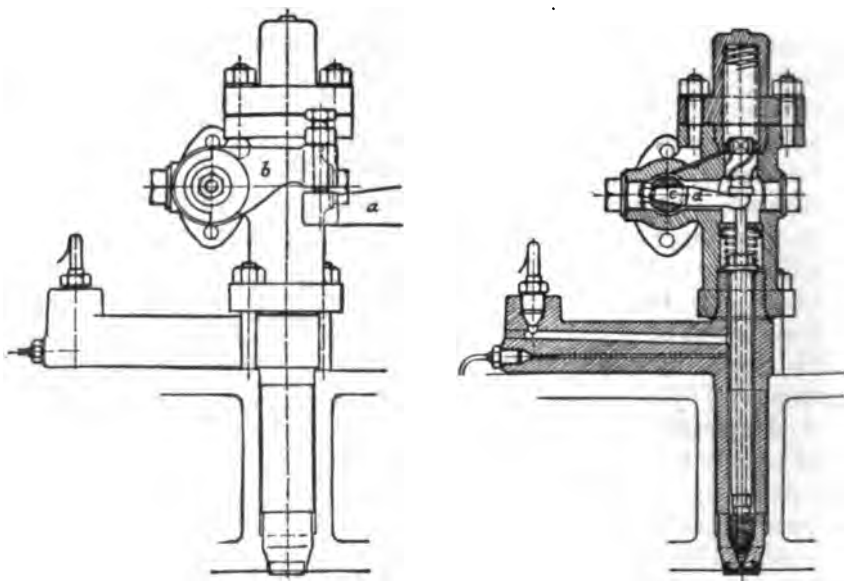


FIGURE 26.—Type of fuel-valve construction permitting easy removal of fuel needle or upper part of valve; a, rocking arm; b, auxiliary lever; c, horizontal pivot joint; d, lever which transmits lifting motion.

products (gas oils and residues) is cheaply obtainable, and coal tar commands relatively higher prices. In Europe, particularly Germany, petroleum residues and gas oils are expensive, and the abundant supply of tar oils from by-product coke ovens and coal-gas works justifies their use by the method outlined. The prospects for the Diesel engine in this field in America are discussed in a subsequent chapter dealing with fuels suitable for the Diesel engine.

METHODS OF REMOVING FUEL NEEDLE.

To remove the fuel needle from the valve it is necessary in some constructions to remove the nut holding the spring bonnet and cap in place, as well as the two nuts that fix the position of the rocker arm through which the needle valve is actuated.

A fuel-valve construction permitting easy and rapid removal of the fuel needle or the upper part of the valve without having to remove the actuating lever, is shown in figure 26. The rocking arm *a* presses upward against a small auxiliary lever *b*, pivoted on a small horizontal joint *c*, to which is secured another parallel lever *d*, which is mounted in the interior of the valve body and transmits the lifting motion to the valve needle. One end of the small journal on which the two parallel levers are fulcrumed is of reduced section and is held fixed by a nut, its function being that of a torsional spring. The opposite end is closed by a stuffing box.

A construction that facilitates the removal of the fuel needle is of decided advantage, as the valve needle has to be removed when it is ground and frequent grinding is necessary. Fuel needles have to be ground weekly or monthly, depending on the quality of the fuel oil. The use of oils with high ash content necessitates more frequent grinding of the needle. The needle is turned back and forth in its seat, a small quantity of emery dust and oil being used. A number of constructions other than that described accomplish the same objects by different means.

Connections of the copper or seamless-steel tubing used for carrying the air and fuel under high pressure are made by joining the ends of the pipes to a copper shank terminating in a cone which fits into a tapered seat machined out of the valve body. A steel gland nut slipped over the copper shank is pressed against the cone to seal the seated connection.

To seal the air chamber of the fuel valve and prevent leakage of air around the valve needle, lead or Babbitt-metal shavings mixed with flaked graphite are used as packing material, secured by appropriate glands. A series of labyrinth grooves on the valve needle constitutes an added precaution against serious leaks.

METHODS OF ACTUATING VALVES.

The time of opening and closing of the different valves of Diesel engines must be accurately controlled; they are therefore not self-acting but are operated mechanically.

SYSTEM OF ROCKING LEVERS AND CAMS.

The usual method of actuating the valves of vertical engines is through a system of rocking levers and cams; the latter are mounted on a horizontal shaft near the top of the cylinders. The cam shaft is driven through a set of helical gears, running in oil, by a vertical shaft, which in turn is driven through another set of helical gears from the engine shaft. The vertical shaft has the same speed as the main shaft; the cam shaft runs at half the speed of the main shaft in engines having a four-stroke cycle and at the same speed as the main shaft

in engines having a two-stroke cycle. The governor, usually of the through-shaft type, is mounted on the vertical shaft, as this has the higher speed. This arrangement permits the use of a smaller governor of the standard type. Governors for horizontal engines are driven from the lay shaft, which also operates the valve gear. This shaft runs at half the speed of the engine shaft in engines having a four-stroke cycle. The governor is therefore speeded up through a set of spur gears, so that a smaller size can be used.

The usual method of actuating the valve gears of four-stroke engines is shown in figure 27. In the figure, a_1 , b_1 , c_1 , and d_1 represent the cams through which the corresponding rocking levers a , b , c , and d , fulcrumed on the short shaft g , are operated. Lever a governs the air-intake valve, b the fuel valve, c the starting valve, and d the exhaust valve.

Levers a and d are operated with shaft g as a fulcrum, with an unchanging center, levers b and c are carried on an eccentric bushing movable around shaft g . Contact between cams and levers is made through rollers, which for the air, starting, and exhaust valves are on the front, and for the fuel valve on the back of the cam shaft e . The arms a , c , and d thus move downward and force open the corresponding valves, whereas arm b has a reversed motion to lift the needle valve. The extent and the duration of the valve lifts are determined by the degree of eccentricity and the length of arc of the cam profile, allowance being made with a small clearance between

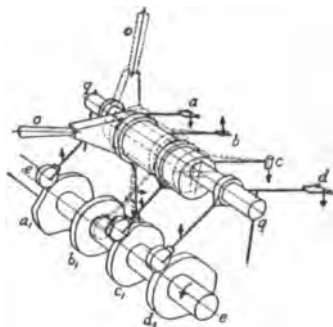


FIGURE 27.—Usual method of actuating valve gears of Diesel engines having four-stroke cycle.

the rollers and the cams during the time the valves are seated (closed). Direction of motion of the different levers is indicated in figure 27 by arrows.

By mounting the rocking arms b and c on an eccentric bushing movable around the shaft g , the following movements are made possible (fig. 28):

1. The engine is at rest, with hand lever o halfway between the vertical and the horizontal position; rocking arm e , actuating the starting valve and rocking arm c , actuating the fuel valve, are removed from contact with the cam. The starting and fuel valves are thus both disengaged. (C, fig. 28.)

2. Handle o is turned to a horizontal position (A, fig. 28), bringing roller e in contact with the cam and engaging the starting valve, roller c being still further removed and keeping the fuel valve disengaged. Opening the valve of the air receiver will admit air through the starting valve and start the engine.

3. When the engine speed required to produce the necessary compression for ignition is obtained, lever *o* is turned to the vertical position (B, fig. 28). Roller *c* is brought into contact with the cam and the fuel valve is engaged, the roller *e* (starting valve) being disengaged the same instant. This is the running position. To stop the engine, the lever *o* is moved into position shown in C, figure 28.

Before these movements are started it is of course necessary to have at least one piston past its top center. Hence the engines are provided with a barring mechanism for turning the cranks into starting position. It is also necessary to pump by hand or by other means to force a small quantity of fuel oil into the pipes leading to the fuel valves, so that the engine will fire quickly.

When the air receivers are placed on the engine-room floor separate from the engine, the operator, if only one is present, will have to mount the engine platform for making the different valve shifts, as well as step down to open or close the valve of the air receiver.

To make possible the performance of this work from the engine-room floor, including the simultaneous engaging or disengaging of all of the starting valves or fuel valves, a system of auxiliary cams is used, controlled centrally from the engine-room floor. This system also permits holding the exhaust valves open during a part of the compression stroke to save compressed air in starting. By shifting the auxiliary cams out of action the regular cams come into play when the engine is running on fuel.

In another construction the air receivers for the air used in starting and injecting the fuel are attached to the front of the engine, being mounted sufficiently high so that the air-receiver valves can be operated and all other movements directed from the engine platform. With this arrangement the accessibility of some parts of the engine suffers somewhat.

AIR-OPERATED PISTON VALVES.

In another construction mechanically operated starting valves are eliminated, independent air-operated piston valves being substituted. These receive the starting air through a rotary distributor operated by the cam shaft. The rotary distributor can be con-

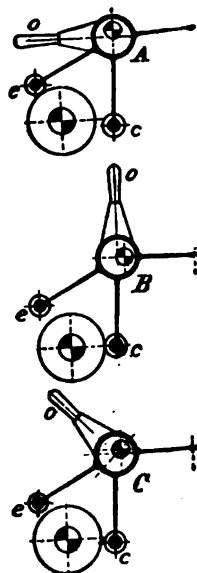


FIGURE 28.—Movements possible by special mounting of rocking arms.

nected or disconnected when the engine is running or is at rest. The fuel valves likewise are thrown in by a central control. This construction eliminates the cam and the fulcrumed lever for each starting valve used with the other types.

As a rule only one or two cylinders of a multicylinder engine are provided with starting valves, thus reducing the cost of the engine. In some engines automatic means are provided for keeping the exhaust valve open during starting, compression being avoided until the engine is well up to or past its normal speed to save compressed air in starting. Some horizontal engines have a blow-off cock in the end of the cylinder, which is kept open for a time in starting; this is also used to blow off accumulations of dirty and spent cylinder oil.

DETAILS OF VALVE-ACTUATING DEVICES.

The rocking-arm levers and the shaft on which they oscillate are carried by individual column stands mounted on each cylinder head.

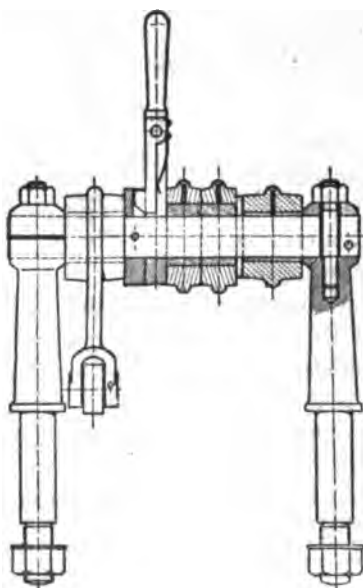


FIGURE 29.—Column stands supporting rocking-arm lever and shaft.

The details of such a stand are shown in figure 29. The shaft (corresponding to shaft *g*, in fig. 27), is fixed in its position with stud bolts. The rocking arms move rapidly and have to resist considerable bending moments; they should be both light and strong. They must also retain their original shape and not be subject to deformation under stress, so as to insure the accuracy of the valve motion. They are therefore made of a good quality of cast steel. The rollers are of hardened steel. The fulcrum bearing is a bronze bushing set in the rocker arm; a ball-and-socket joint insures an articulated contact between the lever and the valve-spring caps.

The cams are of cast steel. They are keyed to the cam shaft, but are usually not set definitely until the engine has been tested. The elements of a gear for operating an air or an exhaust valve are shown in figure 30.

The cam actuating the fuel valve must time the opening of the valve correctly and control the length of time the valve is open; it must therefore be carefully set to insure the best operating performance of the engine. The eccentric is a separate steel inset, se-

cured to the cam wheel with screws. Slotted holes permit the shifting of the eccentric when its correct position has been ascertained from indicator diagrams; it is then locked by small end shims (fig. 31).

As the exhaust valves soon become fouled with carbon, they have to be ground often, and it is desirable that they be readily removable without disturbing the valve-lever operating gear and the fulcrum shaft. This requirement is met by the construction shown in Plate I, B (p. 20). The

lever arm that engages the valve is split and the two parts are joined with two dowel pins and held together by a bolt. By removing the bolt and a part of the lever arm, the entire valve cage can be removed without disturbing the rest of the valve gear. The dowel pins insure the correct realignment of the lever arm. To remove the fuel needle, its rocking arm need not be dismantled by using a construction such as shown in figure 26 (p. 46).

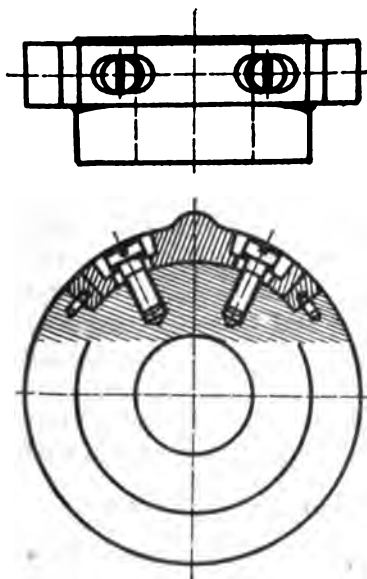


FIGURE 31.—View of cam for actuating fuel valve, showing slotted holes.

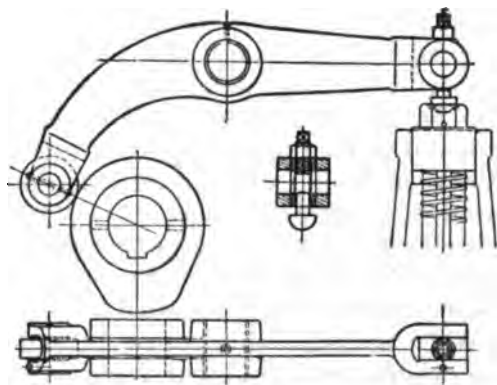


FIGURE 30.—Elements of a gear for operating an air or an exhaust valve.

To avoid long shutdowns of the engine, a spare exhaust valve with cage is kept on hand, which is slipped into the place of the one removed. The frequency with which exhaust valves have to be removed depends greatly on the nature of the fuel burned; fuels with high ash and coke content will foul the exhaust valves more rapidly. Although engines can be operated continuously for several weeks, the usual practice is to stop an engine for about an hour weekly, and remove the exhaust valve and fuel needle of one cylinder.

The valves and needles of the other cylinders are removed in rotation, each week. The valves are removed regardless of their condition merely to keep the engine at its highest efficiency.

Another construction permitting easy removal of valve and needle is shown in Plate VI, *A*. The exhaust and air valve levers are carried by overhanging shaft ends, and can be slipped off by removing a cap bolt and washer, the washer being locked by a small pin set in a hole in the shaft end. In lieu of the two-column stand for supporting the lever shaft, a forked bracket is used, with the rocker arm for the fuel needle in the middle. The needle valve can be removed without disturbing its actuating lever, through an auxiliary system of levers, which reverse the lever-lifting motion. By this means the rocker-arm roller for the fuel needle, in common with the other rocker arms, is engaged by the cam on the front of the cam shaft. The rocker arm for the starting valve is eliminated by the use of a starting valve operated by air.

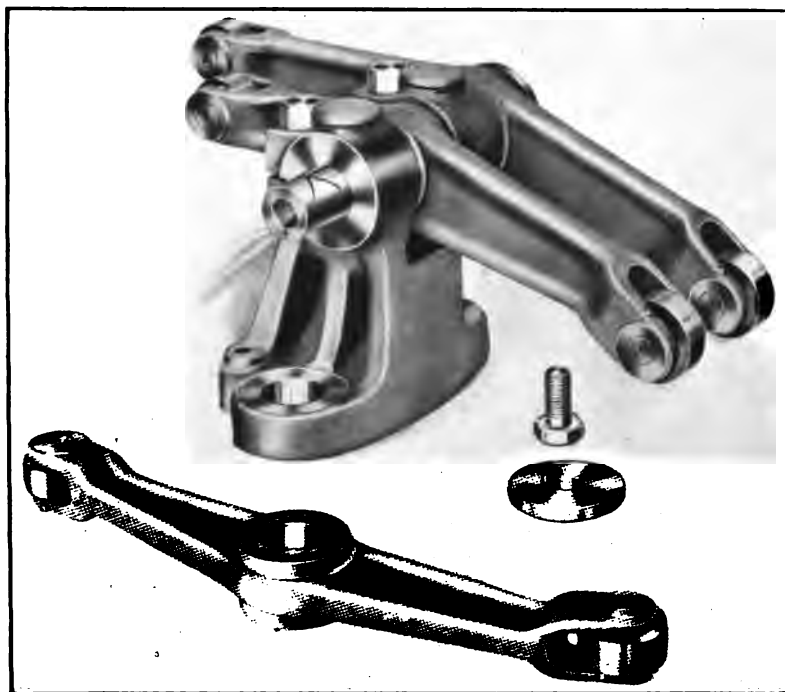
When the scavenging valves of two-cycle engines are placed in the cylinder head, the four valves are operated in pairs; and each yoke (or valve pair) is actuated by a system of double rocking arms, so fulcrumed as to form a parallelogram which serves to actuate all four valves simultaneously and to open each valve an equal amount (see fig. 17, p. 33).

One type of cam shaft is supported by vertical bracket bearings, bolted to facings on the cylinder frames. Ring-oiling cam-shaft bearings are preferable. As the cam surfaces and rocking-arm rollers in contact with each other should be well oiled, some builders house the cam shaft for its entire length and place the cam-shaft bearings in the housing. The housing is carried by brackets cast integral with the cylinder frame and with horizontal machined surfaces. This construction insures a rigid support for the housing and correct alignment of the cam shaft. The cams run in oil, and the oil is carried to the bearings by wipers. Hinged covers allow the inspection of cams, rollers, and bearings. The covers when closed prevent spilling and flying of oil. This construction insures thorough lubrication of all rubbing surfaces, reduces to a minimum friction in the operation of the valve gear, and keeps the engine and the engine room clean. Plate VII shows a cam-shaft housing with housing covers open.

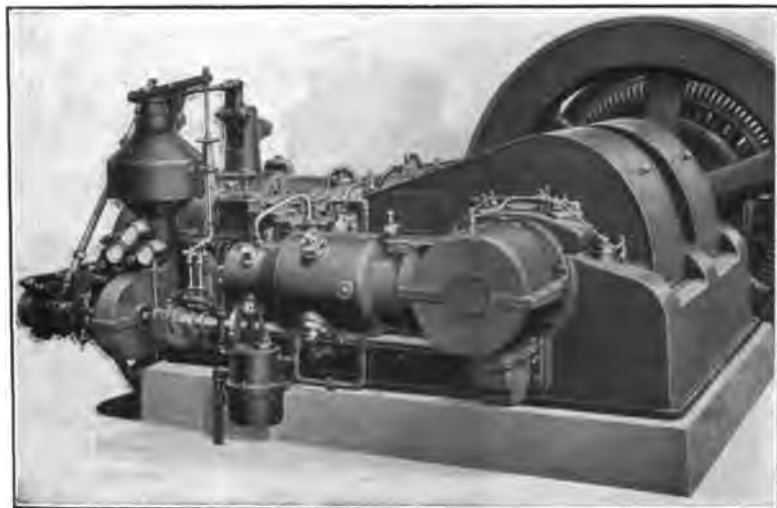
Plate II, *B* (p. 26), shows a method of driving a vertical governor shaft on a vertical engine through helical gears from the main shaft.

VALVE-ACTUATING MECHANISMS IN HORIZONTAL ENGINES.

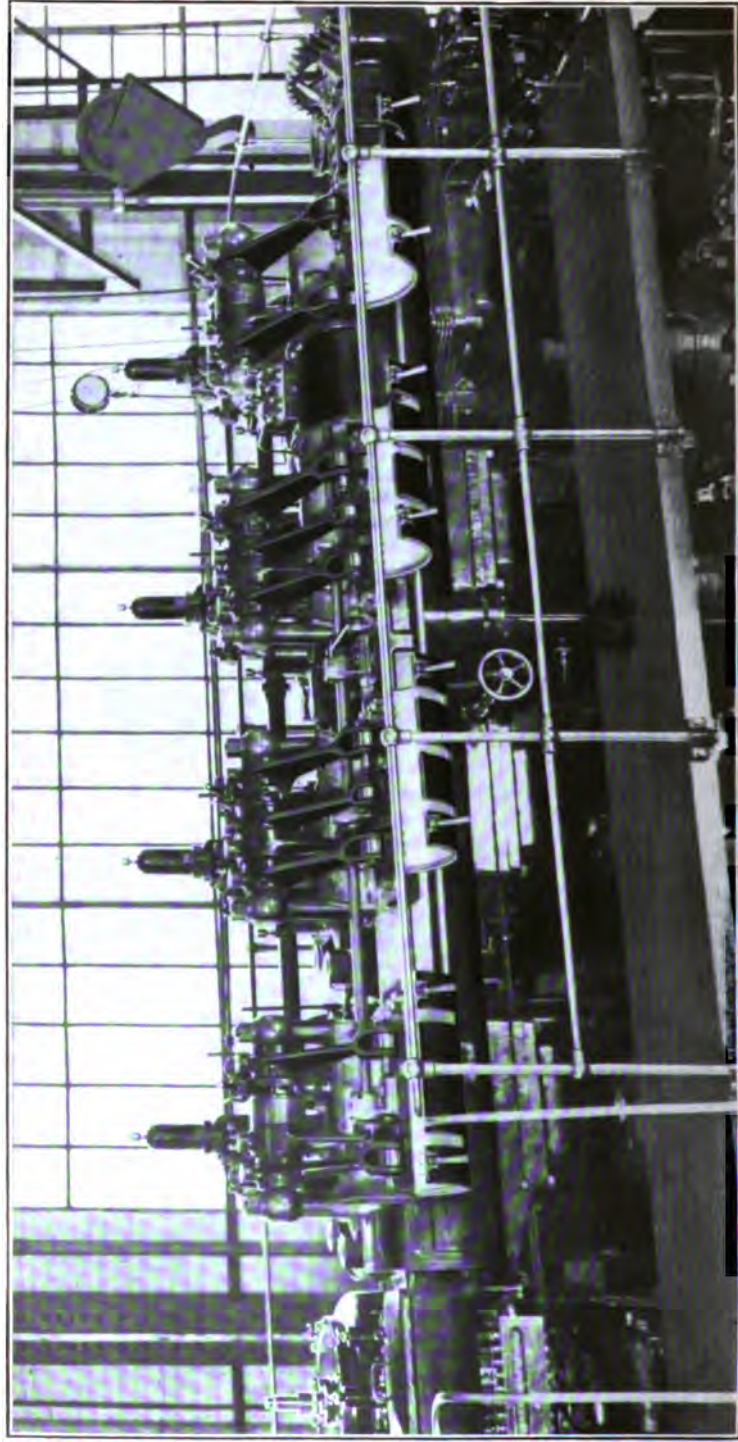
In horizontal engines the governor shaft is driven through helical gears from the engine main shaft, and is mounted horizontally along one side of the engine frame. If the valves are placed in the end of the head, a placing corresponding to the top of the head in vertical engines, the cam shaft is mounted in brackets similar to those of vertical engines. This shaft is driven through a set of bevel gears from the governor shaft.



A. CONSTRUCTION OF FUEL-VALVE LEVERS PERMITTING EASY REMOVAL OF VALVE AND NEEDLE.



B. DIESEL ENGINE BUILT BY ALLIS-CHALMERS MANUFACTURING CO.



DIESEL ENGINE BUILT BY BUSCH-SULZER BROS. DIESEL ENGINE CO. SHOWS CAM-SHAFT HOUSING WITH HOUSING COVERS OPEN.

In horizontal engines having gas-engine heads with the air valves in the top and the exhaust valves in the bottom of the head, the valve levers are operated through eccentrics from the lay shaft. To accelerate the lifting of the valves and to retard their seating, cams with rolling contacts are used. The fuel valve is placed in the center of the head and is operated through a lever driven by an eccentric from the lay shaft, or a separate short shaft is used at right angles to the lay shaft and across the cylinder head, from which the fuel valve is engaged by a cam.

The starting valve or valves are operated by air and are controlled by air-distributing valves operated by a cam on the lay shaft, with which they are brought in or out of contact by a hand lever. Other engines mount the cams on an eccentric sleeve, by the shifting of which the starting valve is engaged, the exhaust valve held open during the compression stroke, and the fuel valve disengaged while the engine is started; shifting this eccentric sleeve to the operating position disengages the starting valve and engages the fuel valve and the other valves in their proper sequence. This arrangement is similar to that used in vertical engines.

The valves of horizontal two-cylinder engines are connected by a system of parallel rods, permitting all valves to be operated from one lay and governor shaft. A separate small shaft across the two cylinder heads driven through a set of helical or bevel gears operates the fuel valve by means of cams. The starting valves are operated in the manner first above described.

The time at which the different valves are opened and closed differs widely with different makes of engines.

The air valves open 15° to 20° before the piston reaches the top dead center, and close 15° to 20° past the bottom center, being open a total period of 210° to 220° .

The fuel valves open 2° to 8° before the piston reaches the top center, and close 18° to 36° after the piston has passed the top center, being open 20° to 44° .

The exhaust valves open 25° to 45° before the piston reaches the bottom center, and close 8° to 14° after the piston has passed the top center.

The angle of lead is least for slow-speed engines and greatest for high-speed marine engines of moderate power (100 to 200 horsepower; 400 to 600 revolutions per minute). This angle makes allowance for the inertia of the valves; the type of fuel valve and the properties of the liquid fuel burned also influence the timing of the fuel valve.

FUEL PUMPS.

The reliability of the fuel pump and its sensitiveness in responding to the slightest variations in load largely determine the satisfactory operation of the engine. The pump must respond to the governor

instantly and must furnish a supply of fuel accurately proportioned to the needs of the engine load. The permissible variations have to deal with extremely small volumes of fuel. As noted above, a fuel volume of 2 c.c. corresponds to a full-load injection for a 50-horsepower cylinder with about 80 injections per minute (160 revolutions per minute). A variation in load of 1 per cent amounts therefore to a fuel volume of only 0.02 c.c., and illustrates how seriously the regulation of an engine may be affected by minute quantities of fuel, unless the fuel pump and governor measure with great precision the fuel required.

As we have distinguished between open-nozzle and closed-nozzle fuel valves, so we may distinguish between pumps that do not have to force the fuel against the high pressure of the injection air but supply the fuel to open nozzles, and pumps that must deliver the oil against the injection air in closed-nozzle valves.

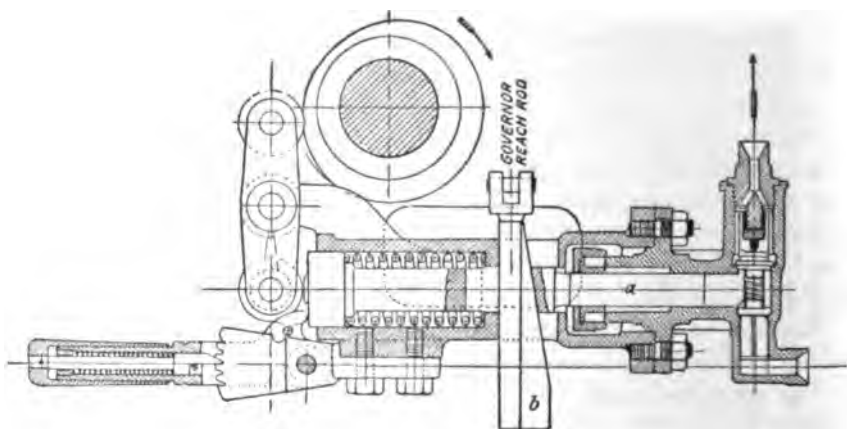


FIGURE 32.—Cross section of oil pump designed to supply oil to open nozzles without opposing air pressure.

PUMPS THAT SUPPLY FUEL TO OPEN NOZZLES.

Pumps of the first type have to work against little resistance—merely that due to pipe friction and the low lift from pump to fuel valve; the oil supply can therefore be regulated with close accuracy by varying the stroke of the pump plunger, which is under governor control.

Figure 32 shows a cross section of a pump of this type. The pump displacement is in proportion to the travel of the plunger *a* which is slotted, a wedge *b*, actuated by the governor moving up and down in this slot, decreasing or increasing the length of stroke and the quantity of oil delivered to the fuel valve.

The cam and the oscillating lever or rocking arm, through which the plunger is actuated, compress a spring, which on its extension reverses the direction of the plunger during the suction stroke of the pump. The plunger can also be operated by the hand lever, before the engine is started, to deliver oil to the fuel valve.

Pumps of this type are simple, work well, and are subject to little wear. When they are working under low pressure, the stuffing boxes are easily kept tight and give no trouble. They respond readily to the governor, and accurately adjust themselves to the smallest load variations.

The fuel pump shown in figure 33 is driven by an eccentric on the lay shaft. The plunger is of the differential type, the upper

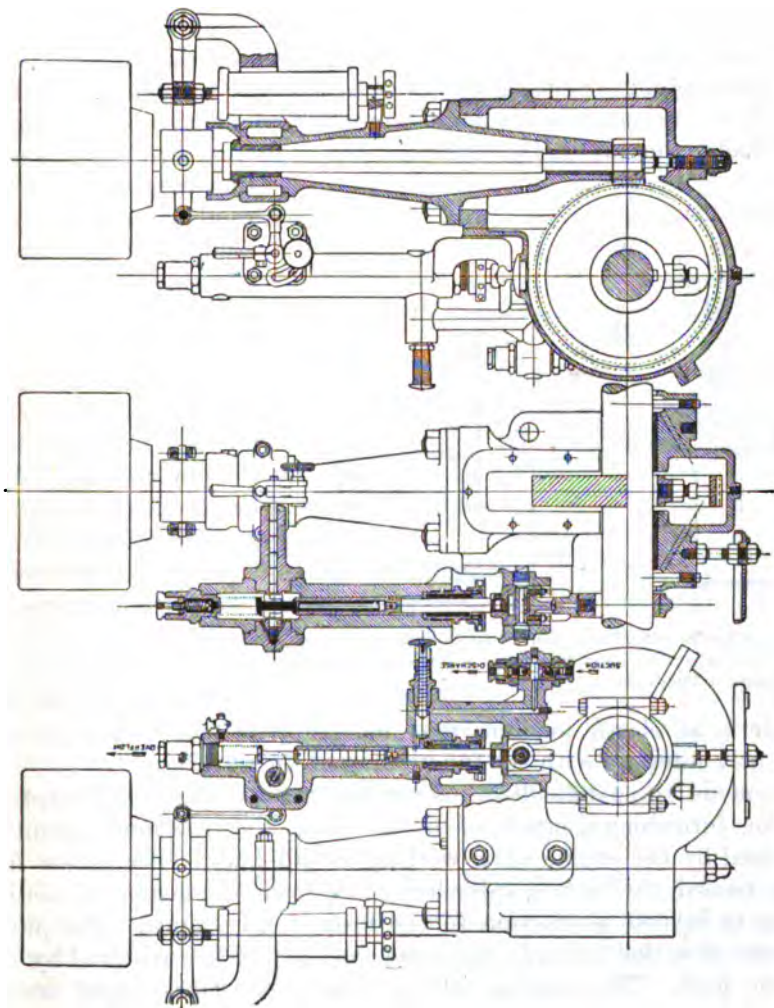


FIGURE 33.—Cross section of oil pump designed to supply oil to open nozzle.

or larger part being hollow and having seated at its upper end a cut-off valve. The plunger has a full positive stroke with each revolution of the lay shaft. The cut-off valve is under direct control of a Jahns governor. Governing is effected by permitting the cut-off valve to seat in a predetermined point in the upstroke of the plunger, thus delivering to the fuel valve a quantity of oil, correctly proportioned to the load that the engine is carrying.

PUMPS DESIGNED TO FORCE FUEL AGAINST HIGH PRESSURE.

Pumps of the other type have to be designed to withstand safely the maximum pressure against which they have to deliver the oil, namely 1,000 pounds per square inch. They have to force the oil into the fuel-valve chamber, which is under the pressure of the injection air that varies from 600 to 1,000 pounds per square inch. Pumps of this type have to deal with extremely dense and correspondingly sluggish liquids.

For this reason no attempt is made, in pumps of this class, to place the pump plunger under governor control and to regulate the fuel-oil supply by varying the stroke of the pump plunger in proportion to the fuel required by the engine. The high pressure under which

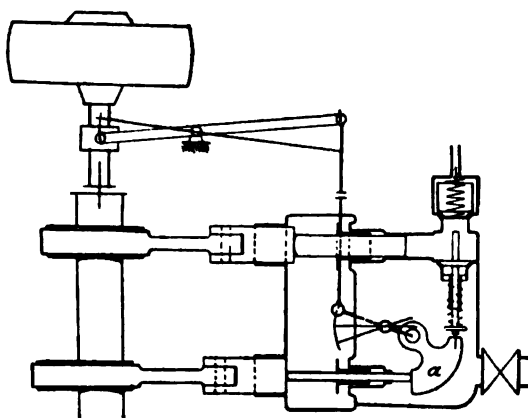


FIGURE 34.—Essential features of oil pump designed to work against pressure of injection air and having constant-stroke plunger and push rod.

the plunger has to work would react on the governor with each stroke, affecting its precision unfavorably and demanding a more powerful governor. There is also the possibility that air may be drawn into the plunger barrel during the suction stroke: with the minute quantities of oil to be delivered to the engine, especially at partial loads, a small air bubble may displace all the oil, and

as air is an elastic medium, may be compressed and expanded and seriously interfere with the functioning of the governor.

To avoid these difficulties the pump plunger works with a constant stroke, furnishing a supply of oil exceeding the maximum quantity required by the engine when working at full load. The excess fuel is by-passed, the suction valve being held open, the degree of opening being in inverse proportion to the load of the engine. The pump plunger thus delivers only the exact quantity of fuel required by the engine load. The suction valve is under governor control and is actuated through a push rod with constant or variable stroke.

The elements of a pump with a constant stroke of the plunger and the push rod are shown in figure 34. The pump is driven by eccentrics from the (vertical) governor shaft. Crank *a* is free to swing around a pin supported by one end of a fulcrumed arm, the other end of which is connected with the governor. The push rod with

constant stroke opens the suction valve by lifting crank *a* around its point of support; the magnitude of the lift is controlled by the extent that *a* is lifted under the influence of the governor. The lower the load the greater is the lift of *a* and, correspondingly, of the suction valve, with a greater quantity of the oil by-passed. Although the stroke of the push rod remains always constant, the surface of crank *a* in contact with the end of the push rod slides past it up or down when being lifted or lowered by the governor.

In other pumps the push rod has a constant stroke, but the starting point of its travel is influenced by the governor, and it subjects the lift of the suction valve to the same variation in travel.

A pump comprising all the elements of the one shown in figure 34, but with a variable-stroke push rod, has crank *a* swinging around a fixed instead of a movable center. The push rod is operated by a variable eccentric device connected with a through-shaft spring governor which shifts the eccentric device relative to the shaft center, and thus alters the stroke of the push rod. The changed lift of crank *a* affects the lift of the suction valve correspondingly.

Pumps of this type are usually provided with a hand crank attached to crank *a*, which permits the lifting of the suction valve to its extreme position and the lifting of the discharge valve sufficiently off its seat to let the oil flow by gravity into the pump chamber and the fuel-valve oil-delivery pipe from the oil-storage tank (which is higher than the engine) by opening a cock connecting the pump chamber with the atmosphere. When the suction valve is held wide open, the engine is stopped. These pumps have been developed to a high degree of perfection, and notwithstanding the severe conditions under which they have to operate they are highly reliable.

Pumps may be vertical or horizontal; the former are used if operated from the cam shaft, although pumps with a variable stroke of the plunger are usually of the horizontal type (fig. 32). Pumps driven from the vertical governor shaft of vertical engines are usually horizontal (fig. 34).

NUMBER OF PUMPS REQUIRED.

Practice regarding the number of fuel pumps supplied with multi-cylinder engines differs, being often decided by a desire to reduce manufacturing costs.

When only one pump is supplied, the fuel has to be divided and proportioned equally to the different cylinders. This is done with small steel diaphragms, the size of the holes being determined experimentally, allowance having to be made for the resistance in the different pipe branches leading to each fuel valve. This method is not entirely satisfactory, especially where a high degree of regulation is desired, as in synchronizing. With a decrease in load, some of the

cylinders will receive an excess of fuel, the supply being proportioned to the load before the governor acted. If one or more of the branches is obstructed, the active cylinder will be overloaded, and if any of the fuel valves or the fuel passages to them leak the active cylinders will be underloaded.

In pumps that are driven by the vertical governor shaft, which, in an engine having a four-stroke cycle, makes double the number of revolutions of the cam shaft, the plunger delivers the fuel in two parts for any one fuel charge, corresponding to one revolution of the cam shaft. The governor, by acting between the two plunger strokes, can therefore adjust the second part of the fuel charge to the new engine load, counteracting somewhat the defect mentioned. As a result of the desire to improve the regulation further, individual pumps are often built as multiplunger pumps, one plunger being provided for each fuel valve (each cylinder). As the plungers are driven simultaneously by one eccentric device, they divide the fuel effectively, but the governor continues to influence only the combined quantity of fuel delivered by all the plungers and not of that delivered by each plunger successively.

To avoid these defects many builders provide an independent pump for each cylinder, each pump being separately under the influence of the governor. The cams or eccentrics actuating these pumps are mounted on the cam shaft (or the governor shaft) with different leading angles, corresponding to the sequence at which the different fuel valves act, and each pump delivers the oil just before the fuel valve it serves opens. Engines so equipped have the most effective regulation.

AIR COMPRESSORS AND AIR RECEIVERS.

GENERAL DETAILS OF AIR COMPRESSORS.

The air required for starting the engine and for atomizing and injecting the fuel is furnished by an air compressor. On account of the high pressure necessary, the air is compressed in two or three stages. Ample cooling of the compressor cylinders and of the air in intercoolers between stages is not merely desirable to approach isothermal compression and reduce the power needed for compression, but is necessary to avoid accidents that may wreck the compressor, with possible injury to the operators. With insufficient cooling of air and excessive use of cylinder-lubricating oil or of an unstable oil, the oil vapor may form an explosive mixture with the heated air and be ignited upon compression. Three-stage compression is therefore much to be preferred, as it permits compressing the air more gradually, and the lower compression ratios avoid excessive terminal temperatures; it also permits cooling the air more thoroughly in intercoolers before it is passed from one stage to the succeeding stage,

The air leaving the last stage of the compressor should be passed through an aftercooler and an oil separator before it is stored in the air receivers. This step is taken not so much to prevent a drop in pressure in the injection air bottle with the contraction of the heated air on cooling as to avoid explosions of the heated air charged with oil vapor. Regrettable accidents have happened from failure to provide intercoolers and aftercoolers. Not only pipes but air receivers as well, have burst. Flame passing through the air pipe and into the air receiver charged with oil vapor has ignited this vapor and blown up the receiver. The aftercooler cools the air, and the oil-and-water vapors are condensed in the separator, and from time to time are drained.

The air compressor should be lubricated thoroughly but without the use of an excessive quantity of oil. The oil used should have a paraffin base, should be of great stability at high temperature, and should have high flash and burning points.

The air compressor is usually driven direct from the main shaft of the engine. On multicylinder engines with closed crank case the compressor usually has the appearance of an additional cylinder, the compressor cylinders and pipe-coil air coolers being surrounded with a mantle forming a water space.

On some engines, especially large units, more than one compressor is used to avoid the large size necessary with only a single compressor. Moreover greater accessibility and reliability are obtained, and the work of compression is divided into a larger number of smaller absolute impulses.

Compressors may be vertical, horizontal, or inclined for both vertical and horizontal engines. They are made of cast iron, with water-jacketed cylinders, or are surrounded by a mantle which forms a large water space and in which are housed the air-cooling coils. More systematic cooling can be done by using water-jacketed cylinders and independent intercoolers. Also, these then become more accessible. The cylinders are stepped down to correspond to the number of stages; likewise the pistons, which are of cast iron. The high-pressure end is usually too small in diameter to permit extending the snap rings sufficiently to slip them over the piston. They are then held between spacing rings and locked with a nut in the end of the piston.

The valves are metal poppet or disk valves. The discharge and intake valves between stages are preferably housed together in one cage, of a construction that facilitates their removal and replacement. The high pressure demands small clearance spaces; the interior of the cylinder-head end should therefore be free of pockets or dead spaces, and the valve seats should conform closely to the inner surface of the cylinder; the piston likewise should conform to the shape of the cylinder head to permit its close approach to the head. Each

intercooler should be fitted with an oil separator, so that excess lubricating oil and condensed water may be separated and drained and not allowed to pass on to the next stage.

A blow-off (safety) valve should be placed in the high-pressure air pipe, leading to the air receiver, directly back of the high-pressure end of the compressor (past its discharge valve), to prevent the bursting of the pipe as the result of an obstruction. The necessity of an aftercooler and an oil separator has already been mentioned.

Compressor troubles are probably responsible for a large proportion of shutdowns in the operation of the Diesel engine. Particular care should be used in compressor construction to make all the parts as simple as possible and still retain the maximum efficiency and reliability. Valves should be readily interchangeable, and the material and workmanship of the highest order.

AIR RECEIVERS FOR AIR COMPRESSOR.

The air for the compressor is usually stored in three air receivers, two large ones for the air used in starting the engine and a smaller one for the injection of air. These receivers are made of seamless drawn steel, and to avoid heavy walls and excessive weight are relatively long and of small diameter ($L/D=4$ to 8). They are interconnected with a system of pipes controlled by valves, so that either of the large bottles may be replenished with air from the injection-air bottle, which is supplied from the air compressor. The valve bodies are machined out of solid forged-steel blocks. The valve bodies are connected to the air pipes by copper cones and steel gland nuts. The valves are controlled by large (8-inch) handwheels.

One of the large air bottles is used for storing a reserve supply of air under a pressure of 1,000 pounds per square inch. Air is drawn from the other when the engine is started. The latter bottle is refilled as soon as the engine is in operation. The air compressor is designed amply large to replenish, in 15 to 20 minutes running, the air so used, without drawing on that used for injecting the fuel with the engine at full load. Manometers are provided to indicate the pressure in the large bottles and the injection-air bottle and the intermediate pressure of the compressor. At least one of the large air receivers is filled with air at the manufacturer's works and shipped with the engine to be used in starting it for the first time. Before the engine is started at any time the operator should convince himself that every part of it is in operating condition; should he fail to start it by turning on the starting air, he should close the air valve at once and locate the cause of failure rather than make a number of attempts and waste the compressed air.

If all the air has been used before the operator has succeeded in starting the engine, which operates the compressor for producing a fresh supply, compressed carbon dioxide (carbonic-acid gas) can be

used in lieu of compressed air. To prevent the freezing of the gas when it is drawn, the top part of the container may be warmed by wrapping cloths soaked in hot water around it. The container should not be heated by such means as torches, as there is great danger of bursting it from overheating the gas.

The size and the number of air receivers vary with the size of the installation. In power plants using a number of engines it is well to interconnect the air receivers of the different engines, so that any one can be drawn upon in emergencies for any of the engines. Small engines need a relatively greater air-storage capacity per horsepower than do large engines. The capacity required varies from 15 to 20 gallons per 100 horsepower for the injection-air bottle and from 60 to 200 gallons per 100 horsepower for the starting-air receivers.

REGULATION OF AIR SUPPLY.

The air supply to the fuel valves is regulated by throttling the discharge valve on the injection-air bottle, and the desired pressure, indicated by a manometer on the injection-air bottle, is maintained by correspondingly throttling the air going to the air compressor at the air intake.

Each bottle is provided with a cock and a drain pipe reaching to the bottom of the bottle to drain it of accumulated condensed water and oil.

For starting some engines low-pressure air is used (100 to 250 pounds per square inch), which is stored in the standard type of sheet-steel, riveted air receivers. The receivers are filled with air drawn from the injection-air bottle and supplied by the air compressor. In other engines the air receivers are eliminated altogether and a seamless steel pipe is used between the compressor and the fuel valves, the pipe taking the place of the injection-air bottle. A separate small compressor driven by a gasoline engine supplies the starting air. This practice lowers the first cost of the engine, but the absence of any air reserve makes it necessary that the compressor furnishing the injection air be of unfailing reliability. With the opening and closing of the fuel valves and the lack of sufficient receiver capacity the pressure fluctuates, and with an obstruction in the pipe the pressure may rise abruptly and burst the pipe, as even the precautionary blow-off valves can not be depended upon to act unfailingly. The presence of a gasoline engine with its highly inflammable fuel introduces a fire risk that is absent in installations that derive the starting air from the engine-driven compressor and have in the engine room only a small supply of fuel oil with a high flash point. This advantage of the Diesel engine should not be appraised too lightly in many industrial plants, such as textile and flour mills.

MATERIAL USED IN AIR PIPES.

The injection-air and fuel pipes subjected to the fuel-pump pressure are made either of seamless-drawn copper or steel. Copper pipes are preferable on account of their greater ductility and uniform strength, even though they have a lower tensile strength than steel pipes. Seamless drawn-steel pipes have failed, owing to local imperfections, where copper pipes have satisfactorily stood the test; but no receiver system or pipes are entirely safe against rupture if the air discharged from the air compressor is not first cooled to remove any entrained oil. For conducting the air used in starting the engine a seamless drawn-steel pipe is used; this pipe is of larger diameter and, as the pressure of air used for starting should not exceed 500 pounds per square inch when admitted to the engine cylinder, is safe. This air is usually at atmospheric temperature and the danger from ignition of explosive mixtures of air and oil vapors is absent.

For circulating cooling water and supplying fuel oil to the fuel-storage tank in the engine room standard black or galvanized-iron pipes are used.

SOURCE AND VOLUME OF AIR USED.

The air for the engine is taken directly from the engine room or from the outside. If the air is taken from the engine room, the customary method is to bolt to the facing of the cylinder head a cast-iron elbow, connecting this with the intake to the air valves; the elbow points downward and is connected with a short length of steel pipe (one elbow and one pipe for each cylinder). Its end is closed with a cap. The air enters through a series of long slits cut into the walls of the pipe. This arrangement lessens the noise and prevents foreign bodies from being drawn into the engine.

As the volume of air drawn into the engine is considerable—about 635 cubic feet per minute for a 100-horsepower engine having a four-stroke cycle and about 850 cubic feet per minute for an engine having a two-stroke cycle—especially in large installations, means of admitting an excess of air to the engine room must be provided. However, the noise will probably be considerable and the windows will vibrate and shake under the continuous waves produced by the periodic air displacements. It is therefore a better plan in large installations to draw the air from the outside through large conduits built in the foundation and connecting with the different air intakes to the valves.

Two-stroke engines are provided with air through the air (scavenging) pump, so it is necessary merely to have the pump intake connected with the air canal.

Wherever the air is likely to be contaminated with dust, it should be filtered to prevent excessive cylinder wear and the shortening of the life of the engine.

EXHAUST PIPES.

The exhaust pipes should be of cast iron as steel pipes corrode rapidly, especially when the fuel oil contains an appreciable quantity of sulphur. The sulphur burns to sulphur dioxide, which may be oxidized to the trioxide in the engine cylinder, and combine with the water vapor of combustion to form sulphurous or sulphuric acid. The exhaust gases should never be cooled to the condensing point of water, which would cause corrosion. As the exhaust gases issuing from the engine are hot (600° to 1,000° F.), the exhaust pipes are jacketed and water cooled in the proximity of the engine to make work around the engine bearable to the attendants and to prevent burns.

It is customary to provide a test cock in the exhaust pipe near the cylinder head. By holding a piece of white paper over this cock determination can easily be made as to whether combustion is perfect or incomplete. The exhaust should be colorless.

STORAGE OF FUEL OIL.

The storing of the fuel oil does not call for any departure from the customary practice in steam-power plants. Closed steel drums set in individual closed arched concrete cellars are somewhat high in first cost, but combine the advantages of low fire risk, easy detection of leaks, and prevention of evaporation loss. They can be filled direct from tank cars. As Diesel plants use about one-third the quantity of fuel oil consumed by efficient steam plants, the storage capacity can be proportionately reduced. The rooms in which the fuel tanks are housed should be properly ventilated to prevent the accumulation of any oil vapor that may escape. Care should be taken not to draw the oil from too near the bottom of the oil tank, to prevent sediment entering the oil-supply pipes.

In the engine room are one or more small fuel-supply tanks, usually of a capacity to run the engine for half a day. In small installations, a hand-operated fuel pump is used to refill the small fuel tank with fuel oil from the main fuel-oil-storage tanks. In larger installations a motor-driven pump is used. The small fuel tanks are supplied with a glass gage to indicate the fuel level. Whenever the viscosity of the oil is such that it will not flow readily through the pipes at ordinary temperatures, means to heat the oil to increase its fluidity sufficiently have to be provided. The usual method is to pass the hot jacket water through pipe coils in the small fuel tanks, or to have these tanks provided with water jackets, through which the hot water from the

engine jacket is passed. Provision then has to be made for another smaller fuel tank, in which gas oil (ignition oil) for starting the engine is stored; this is switched on when the engine is to be stopped for any length of time. The fuel supplied to the pipe leading to the engine fuel pump or pumps is controlled from either supply tank by a three-way cock.

When the fuel oil contains sand or other foreign matter, small filter tanks are provided; the fuel oil from the small fuel-supply tank flows through these filter tanks into the main fuel-supply pipe to the engine fuel pump through two branches (one from each filter tank) controlled by a three-way cock. This arrangement permits switching one tank off the circuit for cleaning while the other continues to supply the engine with fuel. As an additional safeguard, filter plugs are placed ahead of the intake valves of the fuel pumps.

The fuel tanks are supported on brackets or a platform on the wall opposite the engine and high enough above it for the fuel to flow by gravity to the engine fuel pumps. Usually each engine is supplied from its own small fuel-supply tank.

COOLING WATER.

For cooling the compressor and the engine a constant flow of water, adjusted to the engine requirements, is desirable. This is best supplied from a constant-head tank, which should be at least 20 feet above the water inlet to the engine, so that the head of water will overcome any vapor tension of the steam formed within the engine jackets.

In some engines the water is first passed through the aftercooler and the intercooler and the water jackets of the compressor, then through the engine-cylinder jackets, and last through the head; in others independent water connections are used, with branches from the main supply pipe to the compressor and the different engine cylinders. Although the quantity of water required for cooling the engine is so much larger than that needed by the compressor that it is not warmed appreciably by passing through the latter first, an independent water supply to the compressor has much to recommend it, affording easier adjustment of the supply of cooling water to the different cylinders and heads. Separate discharges for the water from the cylinders and from the cylinder head should be provided, each with its own thermometer, so that the temperature in a cylinder or in the head can be controlled. The discharge pipes should be provided with valves, to regulate the discharge, as the water is under pressure.

In large installations, all the water-discharge pipes are brought to a central discharge point and mounted on a switchboard; a cock regulates the overflow, and a thermometer, mounted on the board above each respective discharge, indicates the temperature. A master valve controls the main supply to the engine. When the

engine is to be shut down, only the master valve is closed, the other valves being left in adjustment.

Some care must be used in adjusting the supply of water to the engine needs, and an excess should be avoided, as it will cool the cylinders too much; they will contract, whereas the piston, being hot, will expand. Excessive cooling may thus cause piston seizures.

When the engine is stopped, the water should be continued in circulation for some time, as the heat stored in the piston, the cylinders, and the cylinder head is considerable, being sufficient to bring the water to the boiling point, so that sudden stopping of the circulation of the water might cause the cracking of a cylinder or of the cylinder head. If the water is hard, lime or magnesia salts may be deposited and interfere with the efficient cooling of the cylinder head and the cylinders, set up internal stresses, and lead to cracked heads and liners. When the water is hard, the best policy is to use distilled water, which should be cooled before use. The loss with an efficient cooling system need not be more than 10 per cent of the volume of water needed for cooling—amounting to one-fourth to one-half gallon of water per horsepower-hour.

The volume of cooling water required varies with the size and type of the engine. Engines having a four-stroke cycle use from 2.7 to 4 gallons of cooling water a horsepower-hour, the larger volume corresponding to smaller engines. Two-stroke engines use from 5 to 6 gallons a horsepower-hour. The volume used depends on the initial and the terminal temperature of the water. The figures given are based on an initial temperature of 50° F. and a discharge temperature of 160° F. It is well to plan on not less than 5 gallons of water per horsepower-hour for an engine having a four-stroke cycle, and not less than 8 gallons for an engine having a two-stroke cycle. The capacity of the circulating pump should be 50 to 100 per cent larger, depending on the type of pump used.

MECHANICAL EFFICIENCY OF DIFFERENT TYPES OF DIESEL ENGINES.

The mechanical efficiency of the Diesel engine, E_m , is the net effective power developed in the engine cylinder remaining after the power absorbed by the moving parts of the engine and by frictional resistance has been deducted. The air compressor, furnishing the injection air and usually driven direct from the engine, consumes considerable of the total power of the engine (7 to 15 per cent) and this loss must also be deducted to ascertain the net effective output of the engine. The mechanical efficiency is expressed as the ratio between the effective power of the engine as measured by a brake on the engine shaft (brake horsepower) and the indicated power as determined from indicator diagrams: $E_m = \frac{\text{BHP}}{\text{IHP}}$.

The mechanical efficiency of a Diesel engine is influenced by numerous factors, such as the type and the size of engine, the quality of the material and workmanship, the care given to details in erecting, the lubricating system, including the quantity and the quality of lubricating oil used, the engine speed, and the volume of cooling

water used. If too much cooling water is used, frictional resistance due to cylinder contraction may be greatly increased. It is always higher in new engines, until the different moving parts have worn the rubbing surfaces smooth.

As the internal power and friction of an engine are nearly constant regardless of load, the mechanical efficiency decreases with a decrease in the engine load.

The mechanical efficiency of engines having a two-stroke cycle is lower than that of engines having a four-stroke cycle as, in addition to the power required by the injection air compressor, there is that required by the scavenging pump.

The mechanical efficiencies of four-stroke engines at full load vary from 75 to 82 per cent, 80 per cent being usual for high-grade, low-speed engines of medium and large powers. The engine efficiency, exclusive of the air compressor, is 85 to 90 per cent.

The mechanical efficiency of engines having a two-stroke cycle seldom exceeds 70 per cent and may be as low as 65 per cent in high-speed engines. The

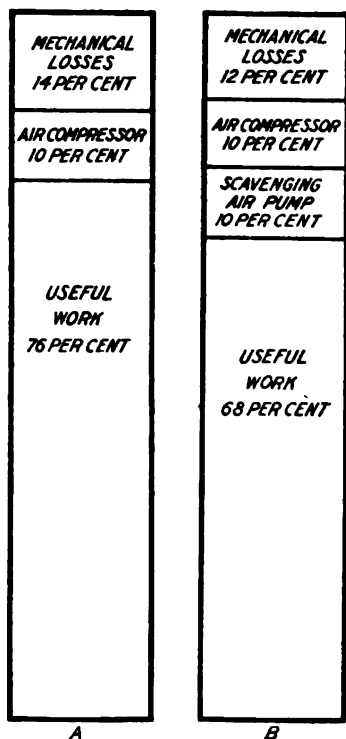


FIGURE 35.—Distribution of power losses in two types of Diesel oil engines: A, engine having four-stroke cycle; B, engine having two-stroke cycle.

distribution of power losses in two-stroke and four-stroke engines is shown in figure 35.

THERMAL EFFICIENCIES.

The thermal efficiency of a Diesel engine is the ratio between the equivalent in heat units of 1 horsepower and the number of heat units actually consumed by the engine in developing 1 horsepower. If based on the indicated horsepower, it is the indicated thermal efficiency, E_{ti} ; if on the brake horsepower, it is effective thermal efficiency, E_{te} .

$$E_{ti} = \frac{550 \times 3600}{778} \times \frac{\text{IHP}}{\text{WH}} = 2545 \frac{\text{IHP}}{\text{WH}}$$

$$E_{te} = \frac{550 \times 3600}{778} \times \frac{\text{BHP}}{\text{WH}} = 2545 \frac{\text{BHP}}{\text{WH}}$$

In the foregoing expressions, 550 foot-pounds per second = 1 horsepower; 778 foot-pounds is the mechanical equivalent of 1 British thermal unit; W is the weight in pounds of the fuel consumed during 1 hour; H is the heating value of the fuel in British thermal units per pound.

If WH is the fuel consumption per horsepower hour, expressed in British thermal units, then $E_t = \frac{2545}{WH}$.

The thermal efficiency depends chiefly on the thermodynamic cycle of the Diesel engine, and is affected by the compression ratio (ratio between total cylinder volume and clearance volume at the end of compression) as well as the cut-off ratio (ratio between cylinder volume at time fuel valve closes and volume at inner dead center of piston or clearance volume).

The indicated thermal efficiency increases with a decrease in the cut-off ratio which contributes to the phenomenal economy of the Diesel engine at fractional loads, the fuel consumption per horsepower-hour remaining nearly constant between full and three-fourths load, and increasing only slightly at one-half load. The ignition of the fuel could be effected at lower pressures, but high compression is essential to high engine economy in Diesel engines.

The mechanical efficiency of the engine naturally influences its fuel economy (thermal efficiency) also, but to a minor degree.

The indicated thermal efficiency of the Diesel engine having a four-stroke cycle varies from 45 per cent at full load to 47 per cent at half load, and the effective thermal efficiency from 37 per cent at full load to 30 per cent at half load, which represents the best practice. As regards engines having a two-stroke cycle, the figures are 10 to 15 per cent lower.

VOLUMETRIC EFFICIENCIES.

The volumetric efficiency (E_v) is the ratio between the weight of a cylinder full of air at the completion of the suction stroke and the weight of a similar volume at standard temperature and pressure. It can be determined by measuring the partial pressure of the air during the suction stroke and dividing it by the atmospheric pressure. The construction of the engine, its piston speed, its valve gear, and temperature are factors that influence the volumetric efficiency. The volumetric efficiency of an engine having a four-stroke cycle differs from that of an engine having a two-stroke cycle. The volumetric efficiency influences the specific duty of the Diesel engine. During the suction stroke of the four-cycle engine, the air becomes somewhat rarified, so that the lower the volumetric efficiency, the lower is the weight of oxygen in a cylinder full of air. The maximum quantity of fuel that can be burned by the air (oxygen) charges is also proportionately lower with lower volumetric efficiency.

The volumetric efficiency of engines having a two-stroke cycle is generally below unity, notwithstanding the fact that the cylinders are filled with slightly compressed air, as it is not possible to scavenge or remove all the gases of combustion, which vitiate the burning power of the air charge. To determine the volumetric efficiency of engines having a two-stroke cycle, it is not sufficient to know the pressure of the air that fills the cylinder (before compression begins); this value must be multiplied by the percentage of pure air present in the total weight of gas filling the cylinder.

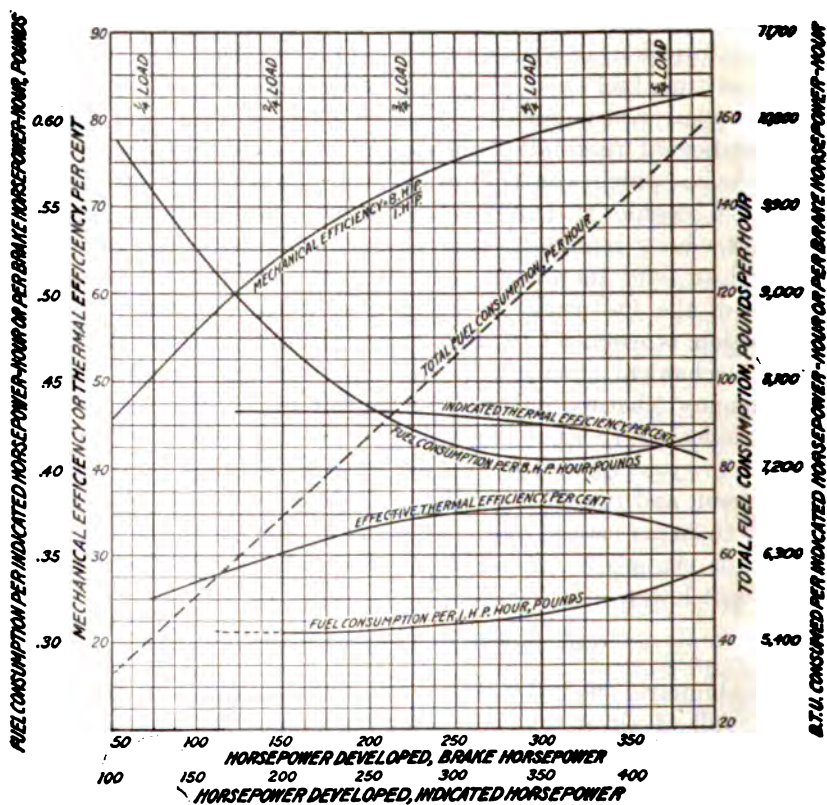


FIGURE 36.—Typical performance curves representing engine efficiencies of 300-horsepower Diesel engines.

For slow-speed four-stroke engines a volumetric efficiency of 90 per cent can be reached, which decreases to 85 per cent for high-speed engines, and for extreme speeds may be even lower. These values presuppose high-grade engines with mechanically operated valves.

Typical performance curves representing engine efficiencies of a 300-horsepower Diesel engine are shown in figure 36.

EFFECT OF HIGH ALTITUDES ON EFFICIENCY.

At higher altitudes the specific duty of Diesel engines decreases appreciably, owing to the lessened density of the air, which affects

the engine just as a low volumetric efficiency would. The horsepower rating of the engine decreases 3 per cent for every 1,000 feet of added altitude. Nearly 40 per cent of the power loss could be recovered by precompressing the rarified air to atmospheric or slightly higher pressure in positive-pressure blowers and filling the engine cylinders with this air. Blower equipment is considerably cheaper per horsepower of capacity than Diesel engine equipment. At high altitudes it may pay, under certain conditions, to install blower equipment for precompressing the air for engines having a four-stroke cycle. An engine having a two-stroke cycle can compress the air readily by using a larger scavenging pump.

CHARACTERISTICS AND USES OF HIGH-SPEED ENGINES.

Diesel engines may be classed as low-speed and high-speed engines. The former are preferred for hard, continuous duty. They have relatively low piston speed, varying from 600 to 800 feet per minute, the piston speed increasing with the power. The number of revolutions per minute varies from 250 to 150, decreasing with the size of the engine. The stroke-bore ratio varies from 1.3 to 1.9, the higher ratio being preferred for low-speed engines for hard service, although with an increase in piston speed the stroke-bore ratio decreases, likewise the number of revolutions.

High-speed engines have a piston speed of 700 to 1,000 feet per minute, a stroke-bore ratio of 1.0 to 1.3, and an engine speed of 250 to 350 revolutions per minute. The speed of engines for special purposes, as for submarines, is often increased to 500 and 600 revolutions per minute with low stroke-bore ratio to obtain a light engine of low height.

The high-speed engines are useful for central-station duty, or for reserve and stand-by units, where the load is relatively light and periodical. The high rotative speed reduces the cost of the generator, makes parallel operation easier, and produces a compact, relatively low engine. The workmanship and materials used in engines of this type have to be first-class, and the materials must be of special quality to be able to withstand the greatly increased specific duty of all the moving parts. Hence engines of this type are by no means reduced in price in proportion to their increased speed, their cost being relatively higher than that of slow-speed engines.

Some manufacturers, to reduce stock sizes, build medium-speed engines only.

The mean effective pressure should not exceed 100 pounds per square inch at full load and is preferably kept around 90 pounds for hard, continuous service. The engines are designed to carry safely momentary overloads of 20 to 25 per cent, when the pressure will go as high as 120 and 125 pounds per square inch. Engines having a two-stroke cycle, in which the number of fuel combustions is generally

double that in engines having a four-stroke cycle, are designed to operate with a lower mean effective pressure—between 65 and 70 pounds per square inch—with a margin for increasing the specific duty of the engine for short periods.

DIESEL ENGINES MADE IN THE UNITED STATES.

The important mechanical features of the Diesel engines manufactured in the United States are briefly summarized below.

ALLIS-CHALMERS MANUFACTURING CO., MILWAUKEE, WIS.

This company builds a horizontal engine with gas-engine head and eccentric-driven admission and exhaust valves. The horizontal com-

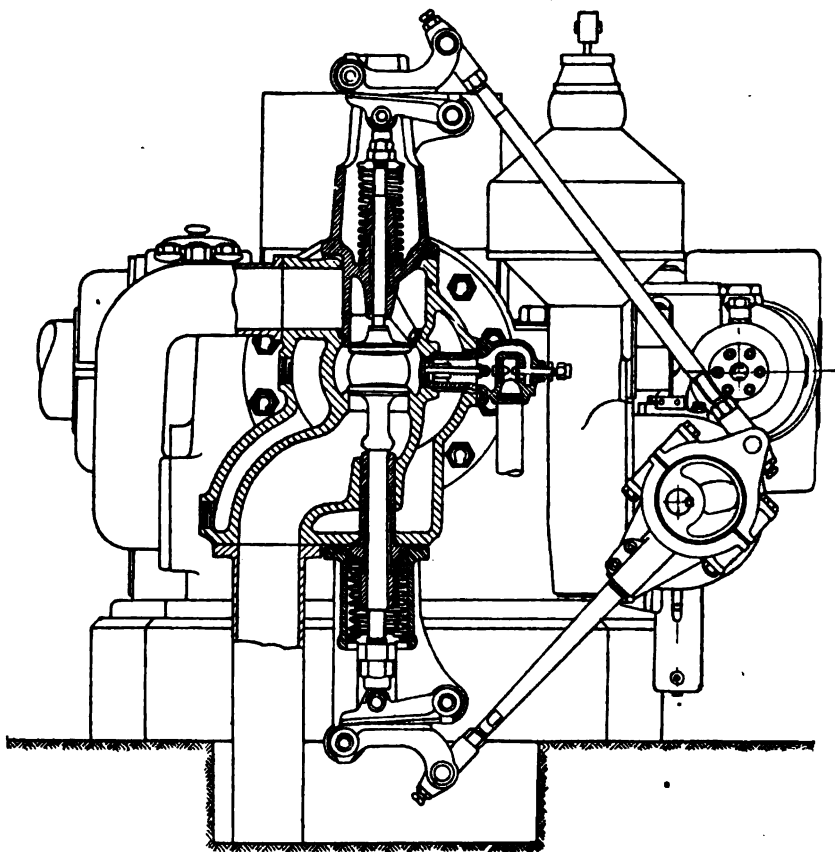
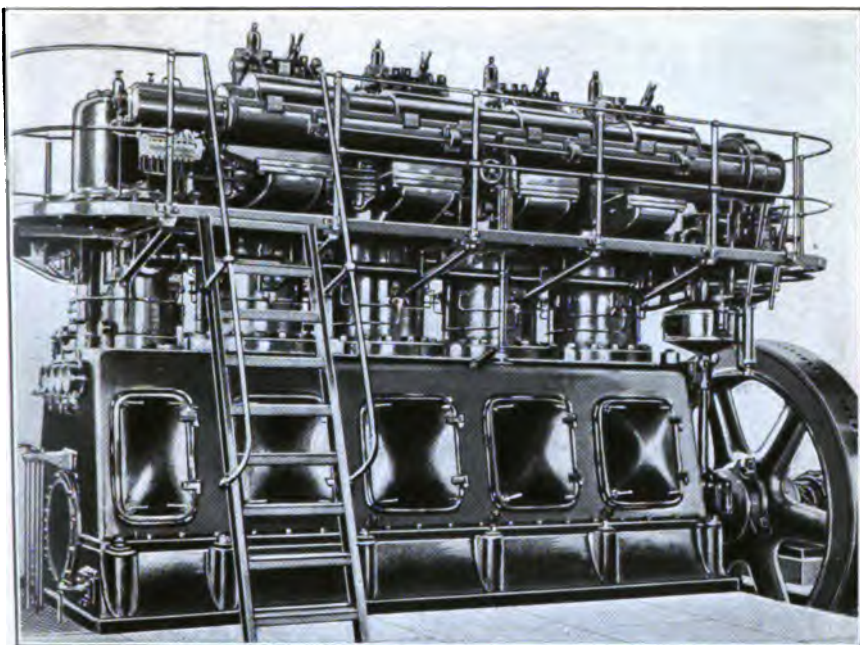
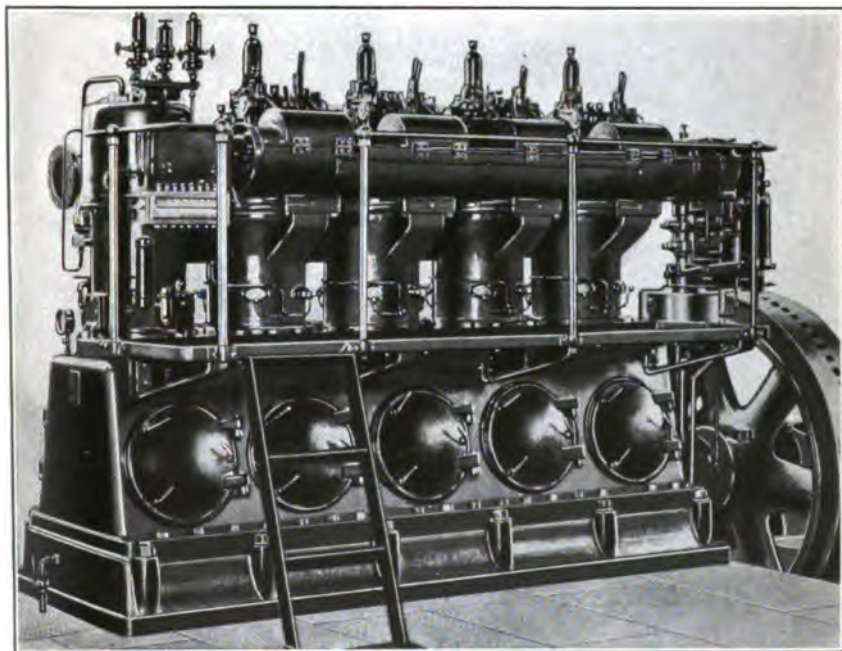


FIGURE 37.—End section of Diesel engine built by Allis-Chalmers Manufacturing Co.

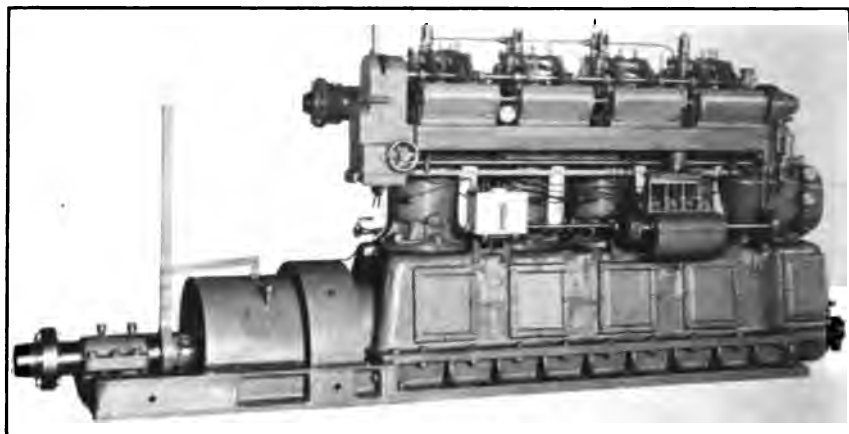
pressor is mounted on the side of the engine frame. It uses the Lietzenmayer system of open fuel nozzle, and variable-stroke plunger fuel pump, which delivers the fuel in the fuel valve during the suction stroke. Figures 37 and 38 and Plate VI, B) p. 52), show views of this engine.



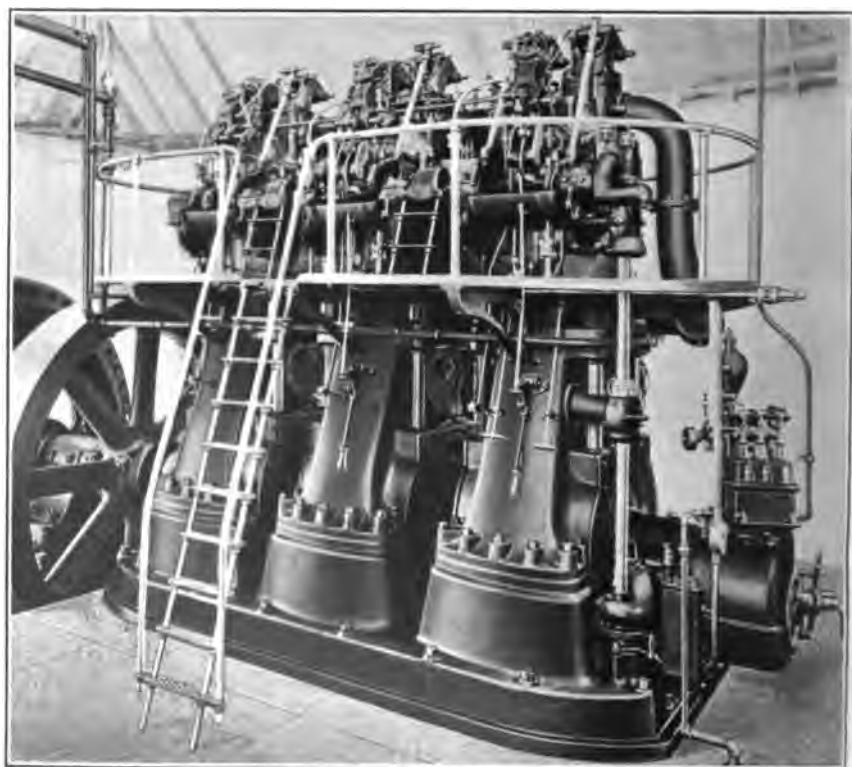
A. BUSCH-SULZER 520-BRAKE-HORSEPOWER DIESEL ENGINE.



B. BUSCH-SULZER 120-BRAKE-HORSEPOWER DIESEL ENGINE.



A. DIESEL ENGINE BUILT BY FULTON MANUFACTURING CO.



B. DIESEL ENGINE BUILT BY LYONS ATLAS CO.

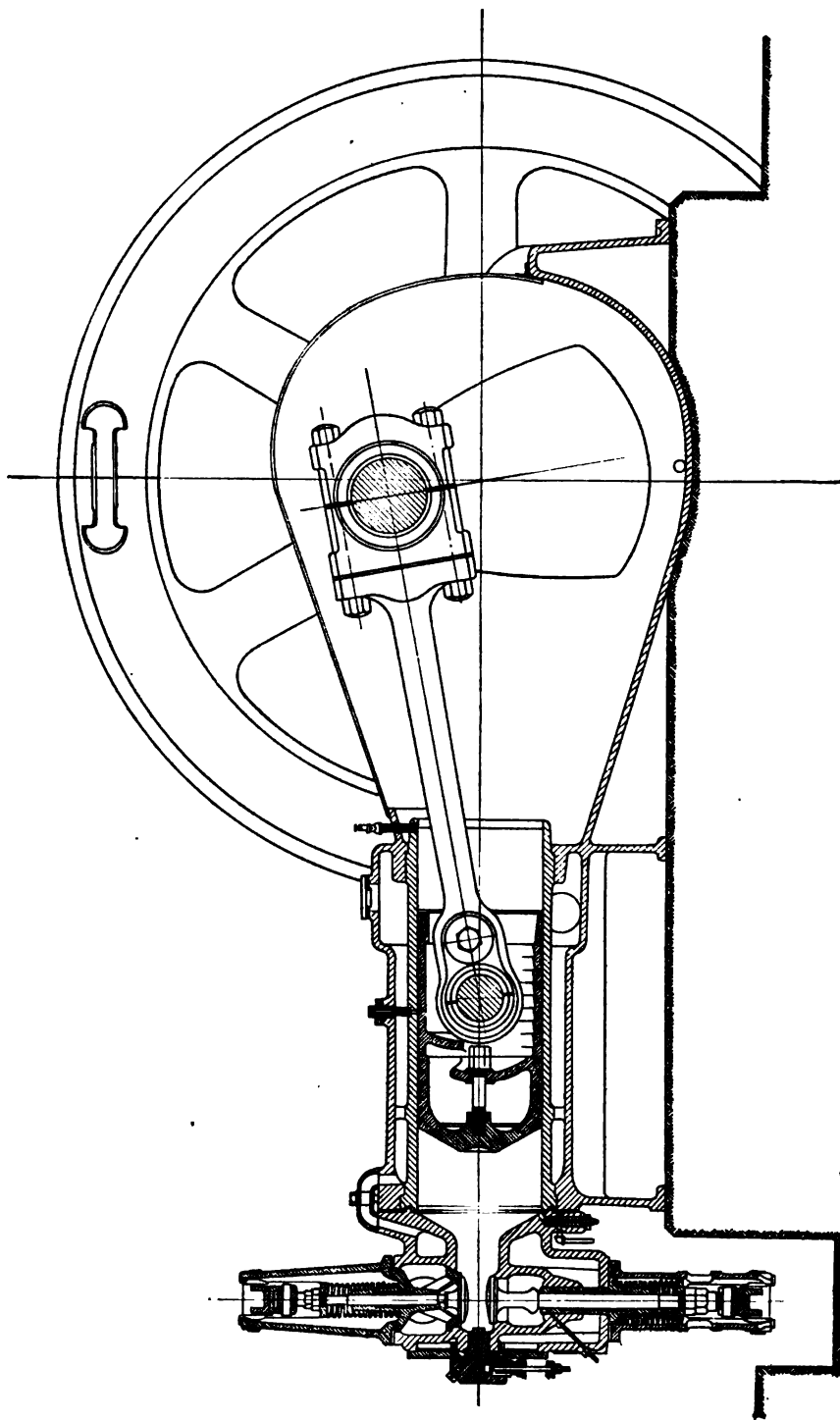


FIGURE 38. — Longitudinal section of Diesel engine built by Allis-Chalmers Manufacturing Co.

BUSCH-SULZER BROS. DIESEL ENGINE CO., ST. LOUIS, MO.

This company was originally the Diesel Motor Co., and later the American Diesel Engine Co., and the first to build Diesel engines in America, since 1898. It still controls a number of the earlier American Diesel patents. It is now associated with Sulzer Bros., of Winterthur, Switzerland. Types of stationary engines placed on the American market by this firm are shown in Plate VIII and in section in figure 39. They are four-cylinder units with closed crank case and four-stroke cycle, the vertical air compressor having the appearance of a fifth cylinder. The first two stages are obtained by a differential and the last by a superimposed piston.

DOW PUMP AND DIESEL ENGINE CO., ALAMEDA, CAL.

This company builds a vertical multicylinder, four-cycle engine, under license from Willans & Robinson, Rugby, England. It is of A-frame construction and conforms to established European Diesel engine practice. The compressor is of the direct-connected, completely inclosed Reavel type.

FULTON IRON WORKS, ST. LOUIS, MO.

This company builds the well-known Tosi engine under license from Franco Tosi, of Milan, Italy. The engine is built as vertical multicylinder units with four-stroke and two-stroke cycle, and with A frames and vertical two-stage and three-stage compressors, depending on the size of the engine. With few modifications, it is built along the lines developed by its European builders.

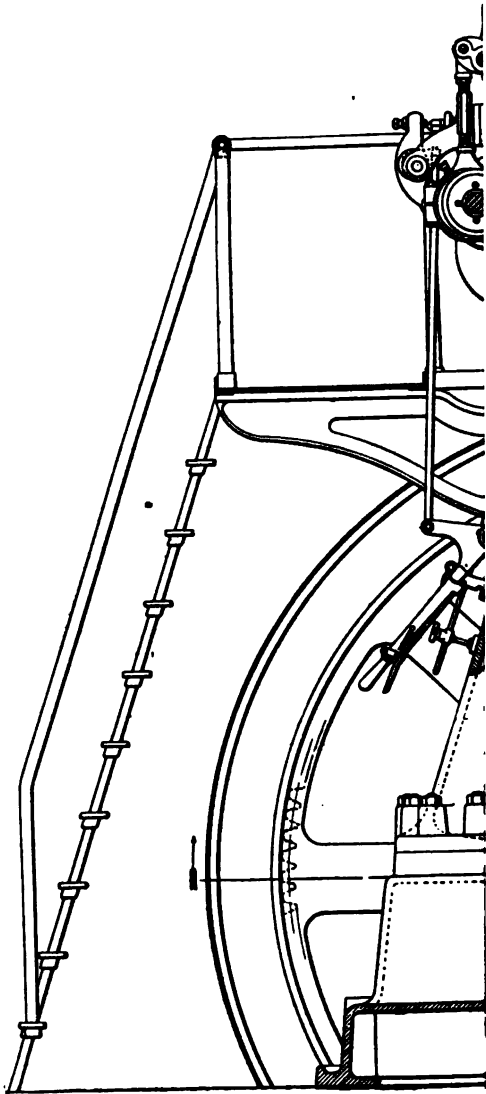
FULTON MANUFACTURING CO., ERIE, PA.

The engine of this company is of the marine type, but can be adapted to stationary work. It is built as a multicylinder engine with four-stroke cycle and with vertical two-stage compressor. It is not directly reversible, but uses a mechanical reversing gear. It is shown in Plate IX, A.

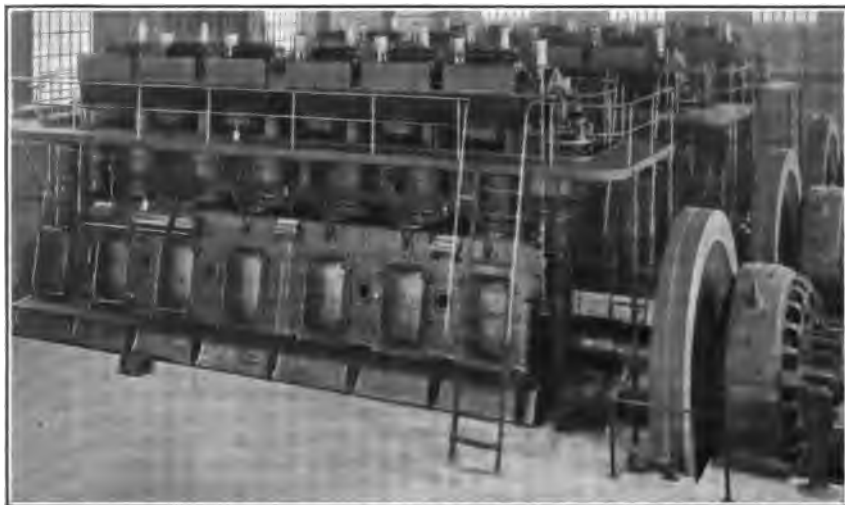
LYONS ATLAS CO., INDIANAPOLIS, IND.

This company developed its own engine, built as two, three, four, and six-cylinder vertical units with four-stroke cycle, all with one standard-sized cylinder of 150 horsepower. The general arrangement of this engine is shown in Plate X. Its general appearance is shown in Plate IX, B.

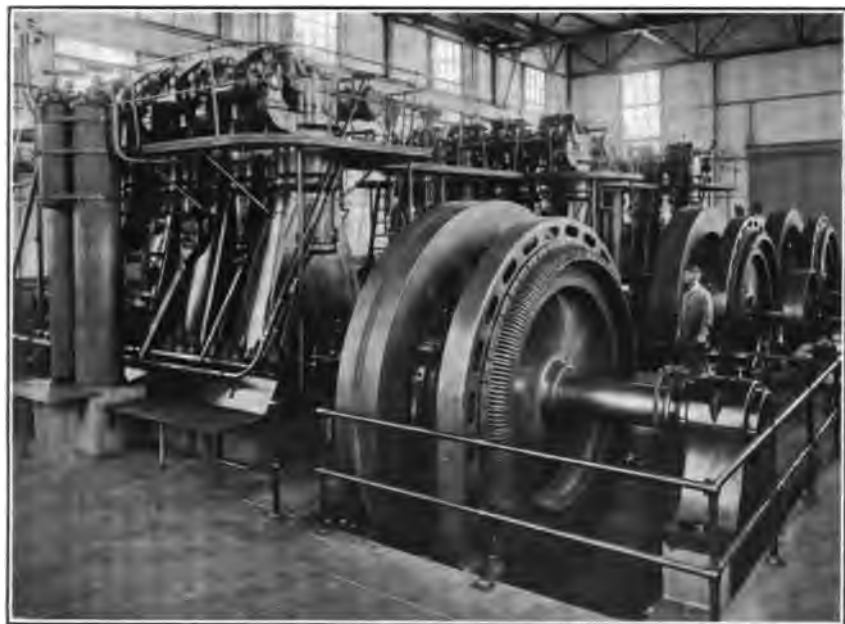
Admission and exhaust valves are operated through eccentrics, actuating wiper cams, and valve levers. The fuel valve is placed in the side of the cylinder head, and is also driven by an eccentric. As the fuel pump shown in figure 34 is controlled by patents owned



CROSS SECTION OF



A. THREE McINTOSH & SEYMOUR 1,000-BRAKE-HORSEPOWER DIESEL ENGINES.



B. THREE McINTOSH & SEYMOUR 500-BRAKE-HORSEPOWER DIESEL ENGINES

by the Busch-Sulzer Bros. Diesel Engine Co., this company designed its own fuel pump. The pump consists of a measuring plunger, whose travel is influenced by the governor. It delivers a measured volume of fuel to the forcing plunger while this is on its downward or suction stroke. Combination check and delivery valves control the flow of oil from one plunger to the other and to the fuel valve. There is a separate set of plungers for each fuel valve (per cylinder), housed in one pump chamber. Injection and starting air is furnished by a separately driven Ingersoll-Rand air compressor.

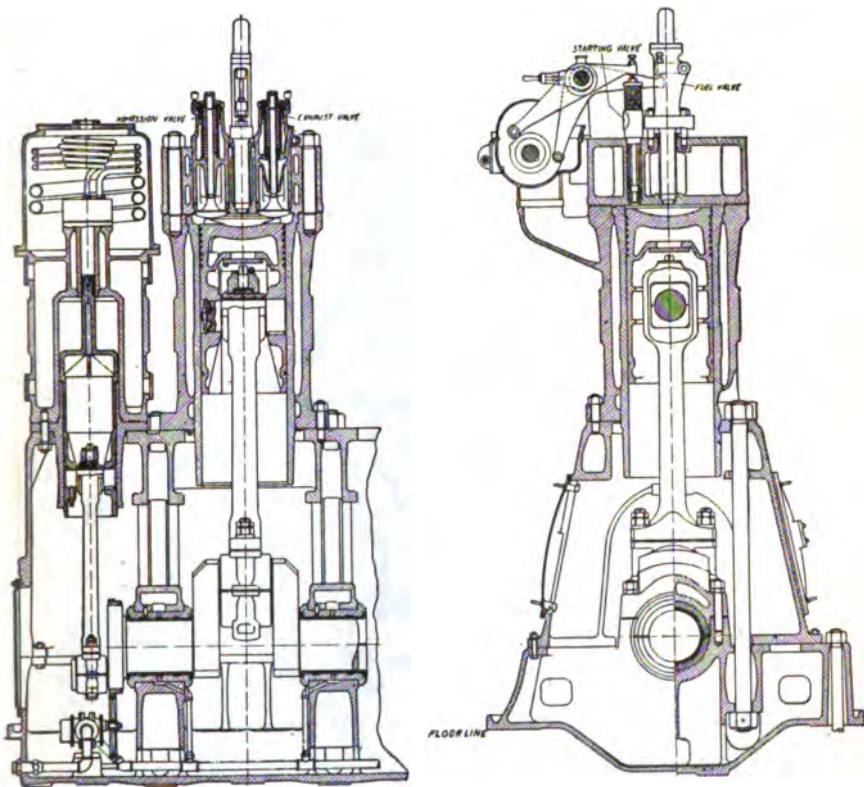


FIGURE 39.—Two sectional views of Diesel engine built by Busch-Sulzer Bros. Diesel Engine Co.

McINTOSH & SEYMOUR CORPORATION, AUBURN, N. Y.

This company is the American licensee of the Swedish Diesel Engine Co. (Aktiebolaget Diesels Motorer, Stockholm, Sweden) and owns the American patents and rights of that company. It builds stationary and marine engines with individual A frames and closed crank cases. They have a four-stroke cycle, and are equipped with two-stage compressors. The company states that its newer engines are fitted with three-stage compressors. The fuel valve developed by Hesselmann is used. In design the engine follows the well-established

lished practice of its European builders. Plants using the engine equipped with A frames and with closed crank case are shown in Plate XI.

NATIONAL TRANSIT PUMP & MACHINE CO., OIL CITY, PA.

The engine of this company is horizontal and is fitted with gas-engine head and rocker and wiper type of valve gear driven by eccentrics from the lay shaft. An open fuel nozzle is used, the injection air valve being operated from the lay shaft by means of a cam. The fuel pump and the governor are mounted together on the side of the main frame. It is driven by an eccentric from the

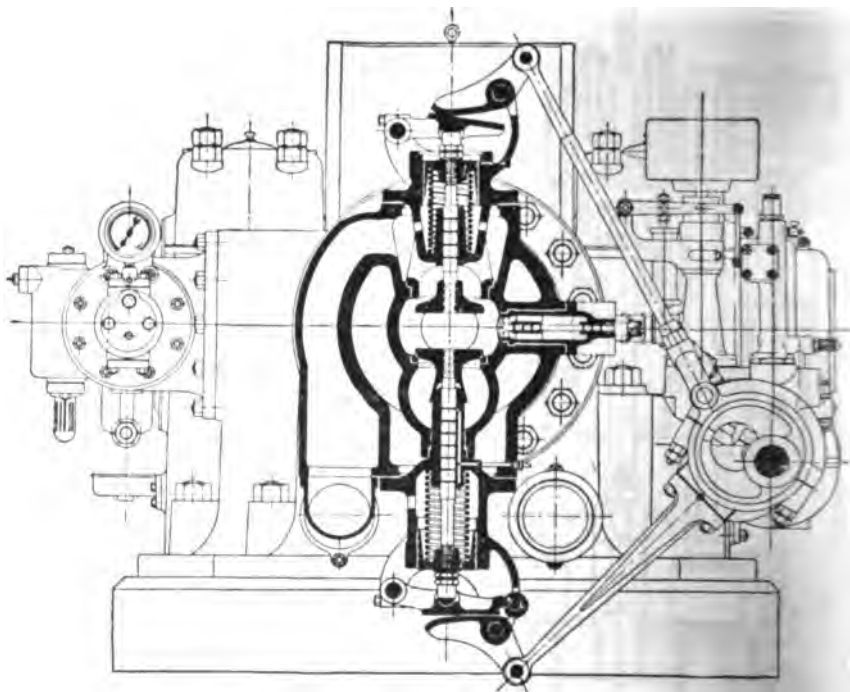
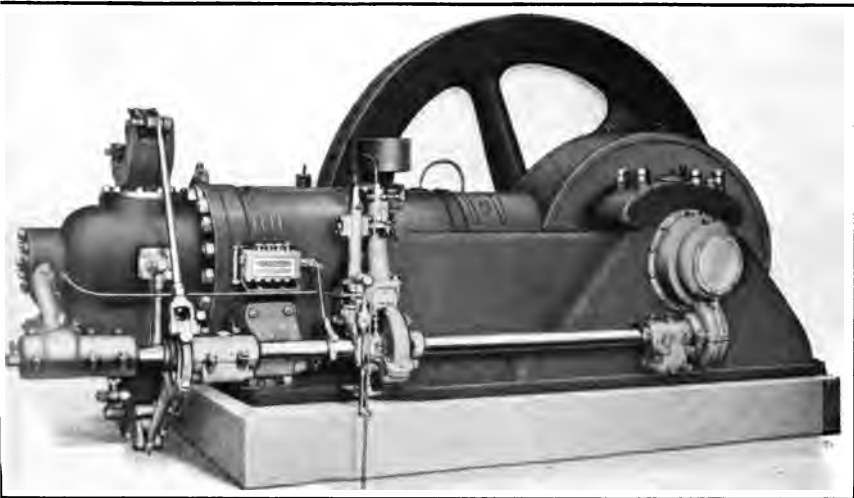
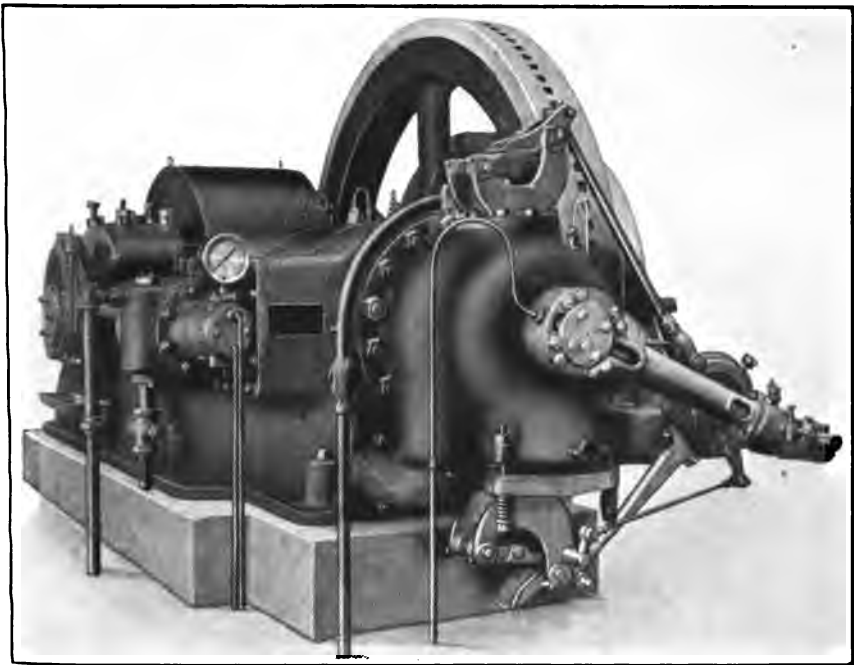


FIGURE 40.—Section through head of engine built by National Transit Pump & Machine Co

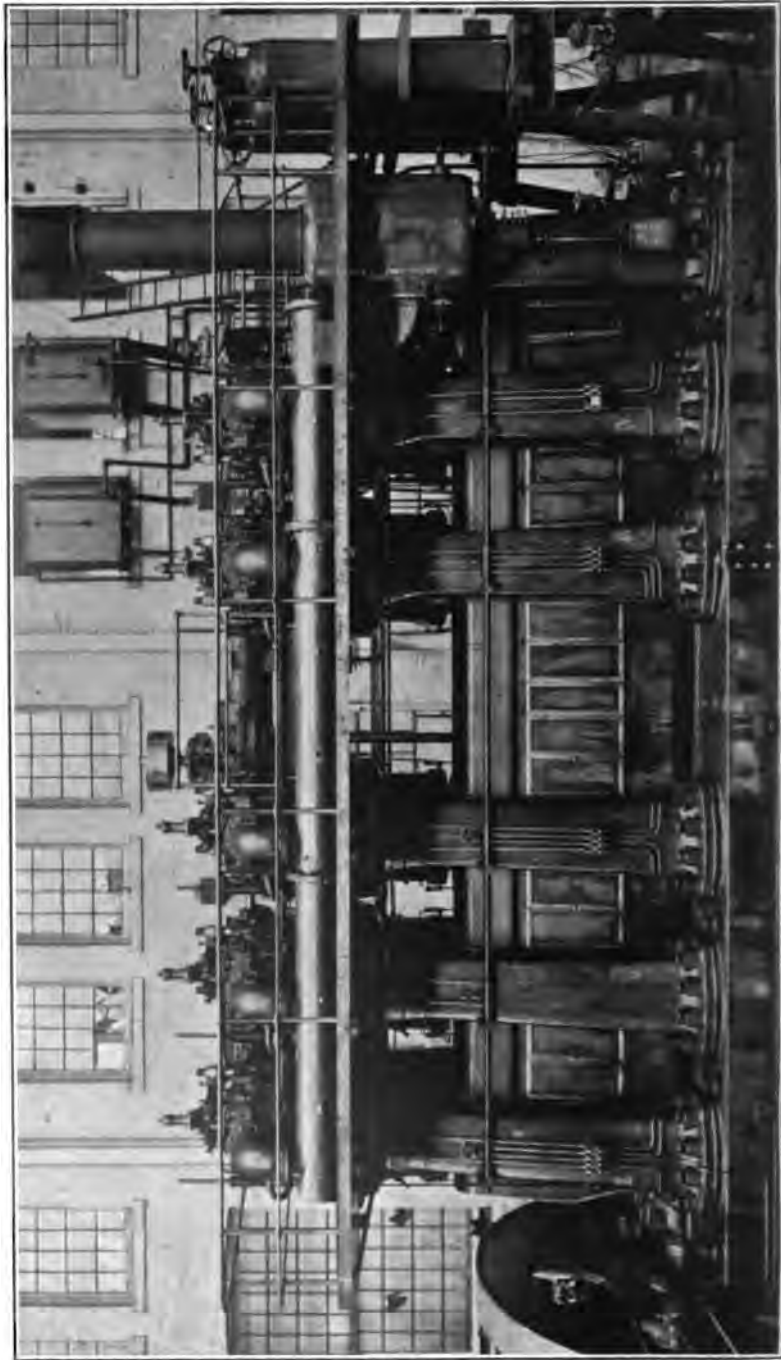
lay shaft. The upper part of the plunger is hollow and has seated on its upper end a cut-off valve which is under direct governor control. The plunger has a constant full stroke, the quantity of oil supplied being proportioned to the engine load. The supply is governed by the distance the cut-off valve is held off its seat, its action being that of a by-pass valve. The cylinder liner is removable. A two-stage air compressor, mounted on the side of the main frame, furnishes the injection air. The engines are built as single and twin cylinder units in sizes from 50 to 350 horsepower. The engine is shown in figures 40 and 41 and in Plate XII. The fuel pump is shown in figure 33 (p. 55).



A. SIDE VIEW OF DIESEL ENGINE BUILT BY NATIONAL TRANSIT PUMP & MACHINE CO.



B. END VIEW OF DIESEL ENGINE BUILT BY NATIONAL TRANSIT PUMP & MACHINE CO.



DIESEL ENGINE BUILT BY NORDBERG MANUFACTURING CO.

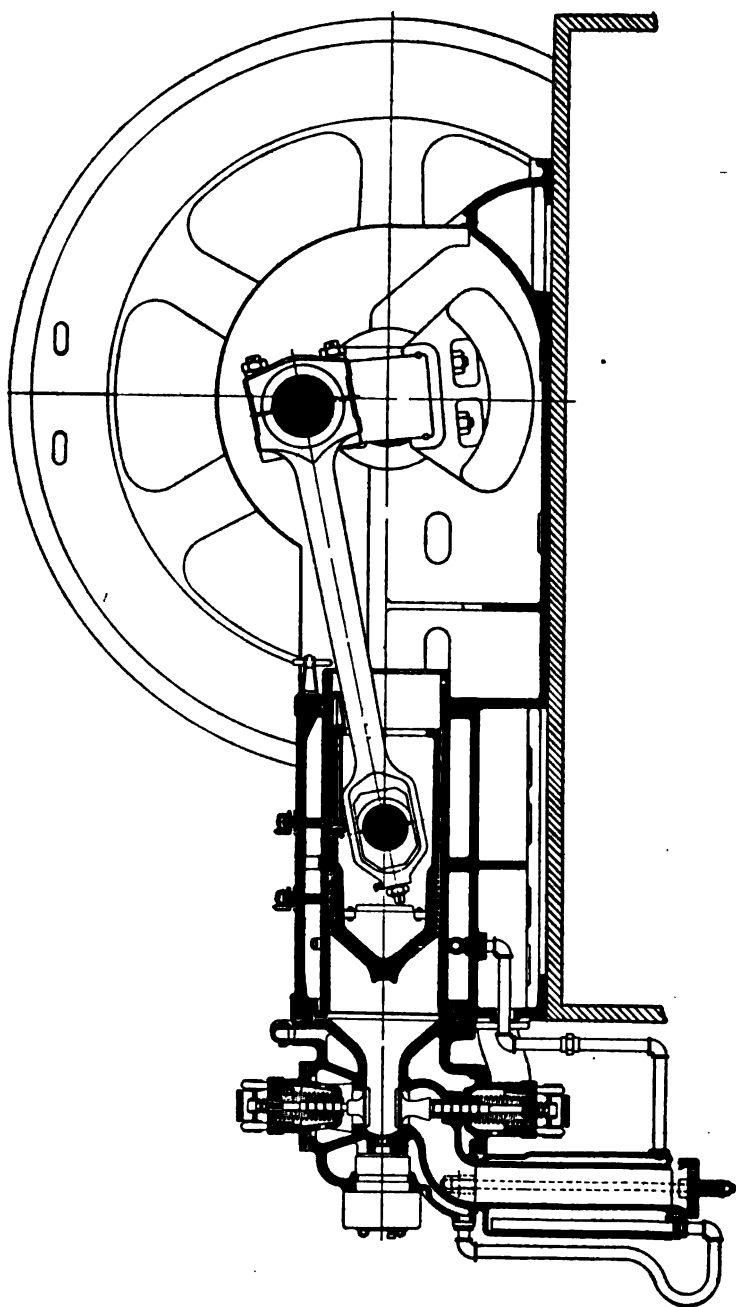


FIGURE 41.—Longitudinal section of engine built by National Transit Pump & Machine Co.

NEW LONDON SHIP & ENGINE CO., NEW LONDON, CONN.

This company builds both stationary and marine engines under license from the Maschinenfabrik, Augsburg, Nurnberg. The stationary engines are horizontal and are built along the lines of the well-known M. A. N. engines of this type.

NORDBERG MANUFACTURING CO., MILWAUKEE, WIS.

This company is the American licensee of Carels Bros., Ghent, Belgium. It builds a single-acting two-stroke engine duplicating the well-known Carels design. Plate XIII shows a five-cylinder engine of this type rated at 1,250-brake horsepower. The scavenging pump is carried by a sixth A frame. Two engines of this type furnished by Carels Bros. have been installed by the Burro Mountain Copper Co., at Tyrone, N. Mex. They are coupled to alternating-current generators and furnish current for the mines and the mills. A third unit like the above and two three-cylinder (750-brake horsepower) engines driving air compressors have been built by the Nordberg Manufacturing Co. for the same plant. A full description of this power plant has been written by Le Grand.^a

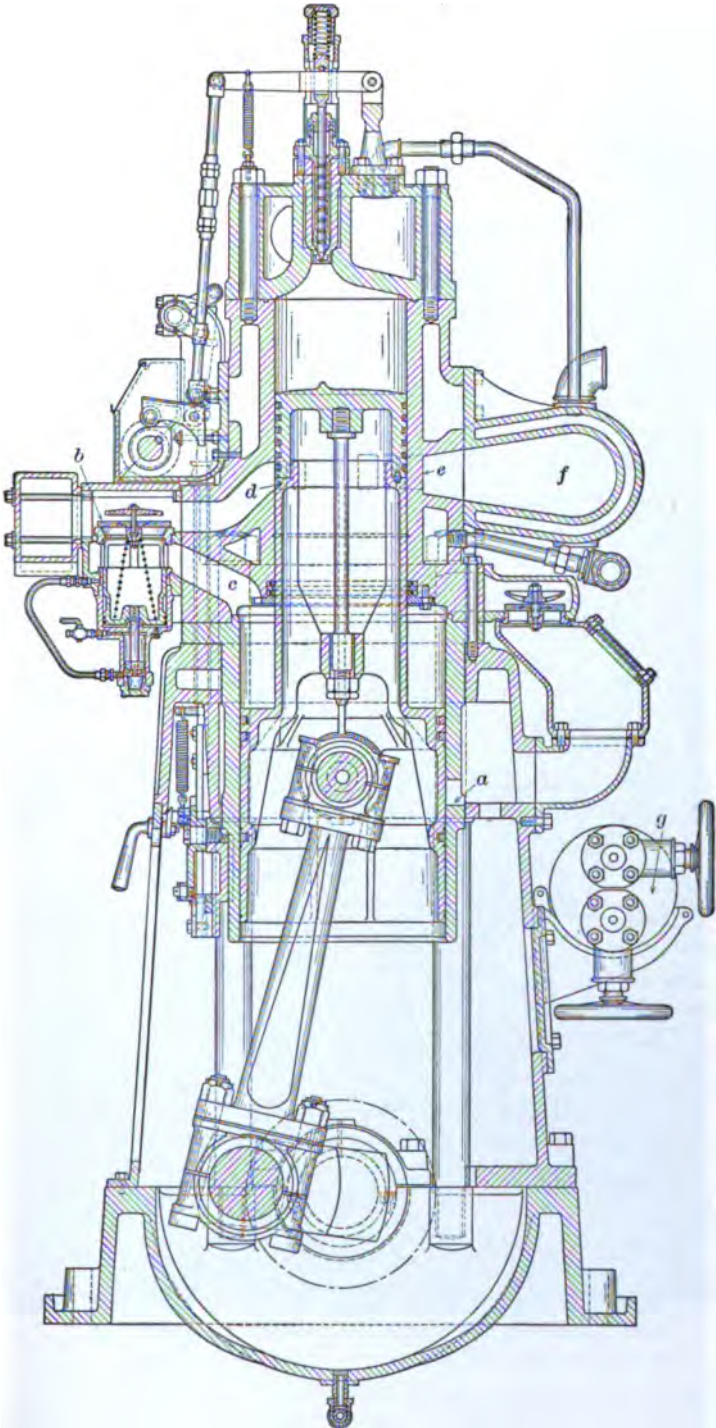
SOUTHWARK FOUNDRY & MACHINERY CO., PHILADELPHIA, PA.

The engine developed by this company is a single-acting two-stroke vertical multicylinder engine used chiefly for marine purposes; it is then built directly reversible. Differential pistons are used in the engine (Pl. XIV), the offset being employed to compress the scavenging air in the lower enlarged part of the cylinder, which thus forms a scavenging pump. Air is admitted into the scavenging pump through ports *a* (Pl. XIV), when these are uncovered by the piston. The scavenging air controlled by valve *b* issues through ports *c*, and when the working cylinder piston uncovers ports *d* the products of combustion are swept out of the cylinder through ports *e* into the water-cooled exhaust main *f*. Only the fuel valves are in the cylinder head. The injection air bottle is shown at *g*. Compressed air for fuel injection for starting and for reversing the engine is furnished by a three-stage compressor. The general appearance of a four-cylinder engine with the air compressor is shown in Plate XV.

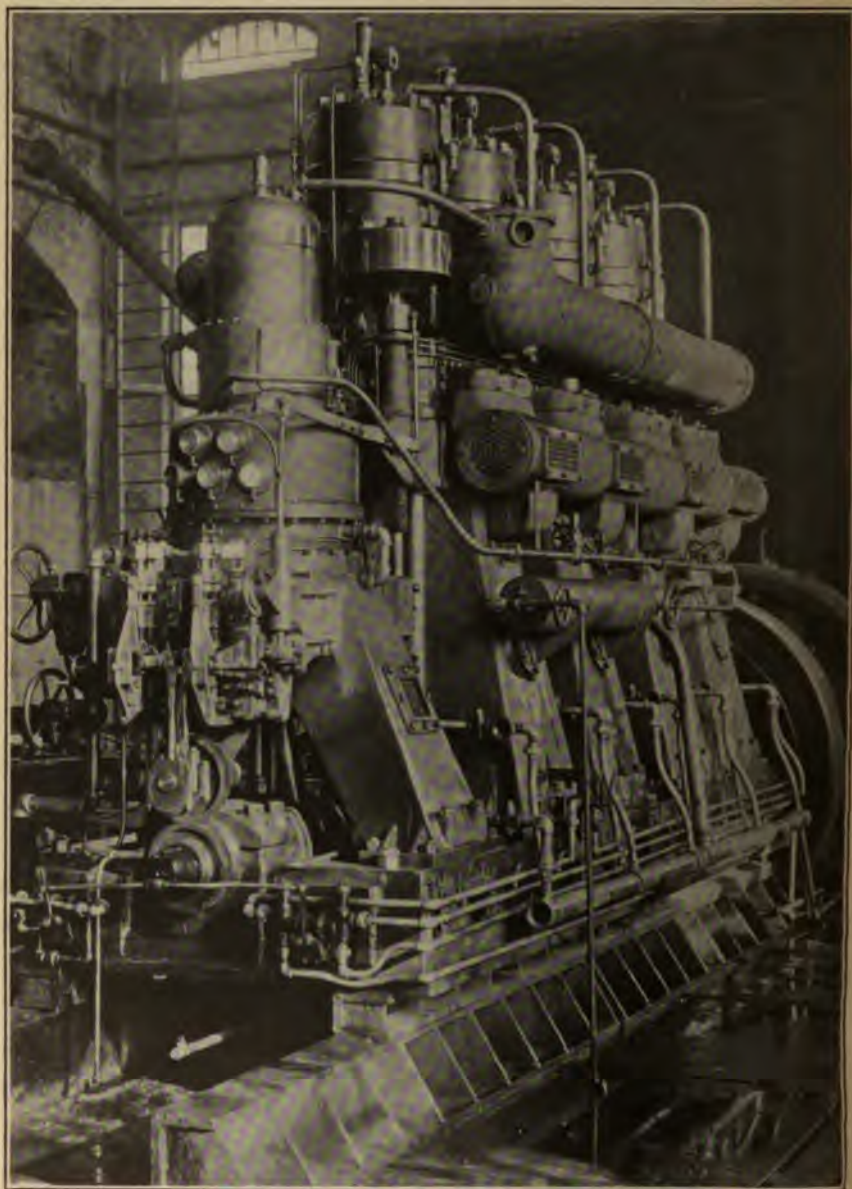
WORTHINGTON PUMP & MACHINERY CORPORATION, NEW YORK, N. Y.

The Snow Steam Pump Works, Buffalo, N. Y., subsidiary of the corporation, builds horizontal, single-acting Diesel engines having two-stroke and four-stroke cycles. Engines with the four-stroke cycle have all valves in the head (front) driven from a cam shaft,

^a Le Grand, Charles, Power plant of the Burro Mountain Copper Co.: Trans. Am. Inst. Min. Eng., vol. 55, September, 1916, pp. 206-217.



VERTICAL SECTION OF DIESEL ENGINE BUILT BY SOUTHWARK FOUNDRY & MACHINERY CO.



DIESEL ENGINE BUILT BY SOUTHWARK FOUNDRY & MACHINERY CO.

which is engaged through miter gears by the governor shaft. The pistons are connected with a crosshead carried in a guide. The compressor is horizontal, is carried on the side of the engine frame, and is driven direct from the main shaft. It uses an open fuel nozzle and variable-stroke plunger pump, which delivers the fuel to the fuel valve during the suction stroke.

LIQUID FUELS FOR DIESEL ENGINES.

CLASSIFICATION OF FUELS.

Liquid fuels for Diesel engines may be divided into three groups, in which the fuels are radically different in structure, chemical constitution, and properties, and accordingly behave differently in the Diesel engine. The first group comprises fuels composed of compounds belonging chiefly to the aliphatic series, namely, a mixture of saturated and unsaturated hydrocarbons, relatively rich in hydrogen, represented by the subgroups C_nH_{2n+2} , C_nH_{2n} , and C_nH_{2n-2} (paraffin or methane, olefiant, and acetylene series). It comprises (1) petroleum and (2) lignite-tar oils. The second group of fuels is composed principally of aromatic hydrocarbons, or benzol derivatives. It comprises (1) coal-tar oils, (2) coal tars, and (3) miscellaneous tars. The third group comprises vegetable oils, which are glycerides of fatty acids.

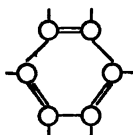
GENERAL CHARACTERISTICS OF DIFFERENT FUELS.

Fuels of the first group are particularly valuable for use in Diesel engines, as they lend themselves to the formation of oil gas at temperatures of 400° to 500° C. The gas is more readily formed and its ignition takes place more readily if it has a high hydrogen content, that is, a proportionately large content of saturated hydrocarbons.

The greater ease with which the oil gas is formed, ignition initiated, and the residual fuel consumed is explained by the chemical structure of these fuels. This structure is best described as being a chain, in which the atoms are held loosely together, and the more inflammable parts, like H and CH_4 , are readily ignited when injected into the heated air of the engine cylinder; with the resulting rise of temperature the required high velocity of flame propagation is obtained as well as complete combustion of the residual fuel particles that do not burn so readily. Carbon particles burn the least readily. Of these fuels the saturated hydrocarbons (paraffin or methane series, C_nH_{2n+2}), have the simplest structure, represented by $\text{---}\bigcirc\text{---}$ methane, CH_4 ; $\text{---}\bigcirc\text{---}\bigcirc\text{---}$ ethane, C_2H_6 ; $\text{---}\bigcirc\text{---}\bigcirc\text{---}\bigcirc\text{---}$ propane, C_3H_8 , etc.

The liquid fuels of the second group mentioned, composed chiefly of aromatic hydrocarbons, are low in hydrogen, are ignited with difficulty, and do not burn readily in Diesel engines. This

characteristic may be explained by their "ring" structure, which holds the atoms together in a closed ring or in combined ring groups of great stability. The ring can be broken or split only by great heat; the oxygen can not attack the "inner" ring of carbon atoms and break it, before the "outer" hydrocarbon radicals, relatively rich in hydrogen, have been ignited and burned and have furnished the high heat required to burn the carbon atoms. The outer hydrocarbon radicals themselves are not readily ignited and split from the inner carbon atoms, as they hold these tenaciously. This behavior is so pronounced that benzol (C_6H_6), which has the simplest structure of the hydrocarbons (which can all be derived from the "benzol ring"



and are also called benzol derivatives), can not be burned in the Diesel engine, and resists combustion far more than the much heavier anthracene and creosote oils, whereas tar oils and tar rich in benzol homologues like toluol, C_6H_5 (CH_3), and xylol, C_6H_4 (CH_3), burn more readily. An explanation for this behavior may be the attack and ignition first of the CH_3 radicals, which, being richest in hydrogen, burn more readily and furnish the requisite heat for burning the rest of the hydrocarbons with high carbon content.

Petroleums rich in saturated hydrocarbons (paraffin base oils) and having the highest hydrogen content burn most readily; with a decreasing hydrogen content, an increasing resistance to rapid oxidation (combustion) is noted. Asphaltic oils and those composed chiefly of naphthenes (heavy California oils) do not burn as readily as the former type.

A product typical of the transition involved in making aromatic hydrocarbons from lignite tar oils, composed chiefly of hydrocarbons belonging to the aliphatic series, is the tar oil made from Scotch bog-head shales and cannel coals. Although many of these liquid fuels have a high residual content (pitch) of about 50 per cent left at $400^\circ C.$, they burn well in the Diesel engine and leave no residue. These tars are the product of a slow distillation at low temperature and differ in physical and chemical characteristics from ordinary coal tars. According to Lunge and Köhler^a they contain, besides paraffins, toluol and naphthalene, but contain little benzol and anthracene. Lacking the most valuable constituent of coal tars, benzol, they have therefore small value as a coal tar, but are valuable as a fuel for Diesel engines.

^a Lunge, Georg, and Köhler, Hippolyt, *Die Industrie des Steinkohlenteers und des Ammoniaks*, Braunschweig, 1912, Ed. 1 and 2.

The resistance to ignition and burning increases still further with the coal-tar oils and is greatest with coal tars, which, with certain exceptions, are not suitable for Diesel engine fuels.

ALIPHATIC HYDROCARBONS.

PETROLEUMS.

Petroleum and certain of its products are the most valuable and abundant source of fuel for Diesel engines, and the United States is fortunate in possessing large petroleum resources; the world's production of petroleum is in round figures about 60 million short tons, of which the United States produces about two-thirds.

DESIRABLE PROPERTIES OF PETROLEUM FUEL.

A desirable petroleum fuel for Diesel engines should have the following properties:

1. It should burn completely without leaving any residual matter in the cylinder, either in the form of soot, coke, or ash.
2. It should be free from mechanical impurities which might clog the fuel pipes, the valves of the fuel pump, and the fine fuel passages in the fuel-injection valves and nozzles, or might cause excessive cylinder wear.
3. It should be sufficiently fluid at ordinary temperatures to flow readily to the fuel pump and thence to the fuel-injection valve.
4. It should be free from water, as water lowers the heating value of the oil and may prevent its ignition.
5. It should be free from highly volatile oils, which will evaporate at ordinary temperatures and form an inflammable mixture with the air, thus introducing a fire hazard.
6. It should have a high heating value.

As is shown later, complete burning of an oil so that no combustible or carbonaceous residues remain in the cylinder, the exhaust being colorless, depends almost entirely on the type and the size of the engine, the fuel-valve and the atomizer construction, and the correct setting of the valves.

TESTS FOR SUITABILITY.

To determine the suitability of petroleum products as fuels for Diesel engines, they should be classified by tests as to the following properties, stated in the order of their importance:

1. Boiling-point range.
2. Ash content.
3. Mechanical impurities.
4. Heating value.
5. Water content.
6. Coke content.

7. Asphalt content.
8. Paraffin content.
9. Sulphur content.
10. Acidity.
11. Elementary composition.
12. Viscosity.
13. Flash point.
14. Burning point.
15. Specific gravity.

FRACTIONATION OR BOILING-POINT TESTS.

Fractionation or boiling-point tests are the most valuable, as they indicate the degree to which an oil can be vaporized, the volume of oil gas formed, and the combustibility of the oil.

The distillation is conducted at atmospheric pressure between the following temperature ranges: 0° to 150° C.; 150° to 200° C.; 200° to 250° C.; 250° to 300° C.; 300° to 350° C.; and 350° to 400° C.

As a general rule the smaller the proportion of residue remaining at a temperature higher than 400° C., and the greater the volume of vapor coming over between 200° and 400° C., the better is the oil suited for use in Diesel engines. A further valuable criterion of the burning qualities of oils that leave residues or "oil tars" at temperatures higher than 400° C. is the quantity of coke left on distilling the residue at temperatures higher than 400° C. The greater the quantity of coke the less suitable is the fuel. Fuel oils with a coke content can not be used in all engines, and a content of 5 per cent may be taken as the upper limit for all engines.

It has been noted that some crude oils that have a high proportion of constituents that produce coke and asphalt and that yield an unusually large proportion of residues at temperatures higher than 400° C. decompose (crack) at temperatures between 300° C. and 400° C. This feature is particularly noticeable in certain California oils and Mexican oils. Such an oil is unsatisfactory for Diesel engines, as only a comparatively small part of the oil is readily burned, a large part being changed to coke; the coke particles contaminate the film of lubricating oil, fill the space between the piston rings, and cover the piston, making lubrication ineffectual, causing increased cylinder wear, and requiring frequent cleaning of the engine.

The distillation test is an index of the fire hazard of an oil. If the volume of vapors passing over at 150° C. (0° to 150° C.) is relatively large, the presence of volatile oils is indicated. It is then well to make another range test between 0° and 50° C. to determine the percentage of highly volatile oils, which affords a better indication of the fire hazard of an oil than even its flash point.

EFFECTS OF ASH CONTENT.

Ash is the most detrimental remnant of the burning of fuel oils for Diesel engines, as it causes excessive wear of cylinders and exhaust valves. The ash is usually composed of mineral particles of great hardness, such as quartz and silicates, or oxides of iron and aluminum, which, becoming mixed with the film of lubricating oil, adhere to the piston and the cylinder walls, accumulate, and cause excessive wear. An ash content in excess of 0.05 per cent will render an otherwise excellent fuel unsuitable for use in a Diesel engine.

MECHANICAL IMPURITIES.

The oil should be free from all mechanically held impurities of an organic or an inorganic nature. When present they should be carefully removed by filtering. Clean oils are sometimes contaminated with impurities when loaded into tank cars in which impurities or dirt have been allowed to accumulate. Impurities also may be introduced by drawing the oil from near the bottom of oil-storage reservoirs or tanks of refineries in which any sediment in the oil may have accumulated at the bottom for a long period. Likewise, at the power plant, the oil should never be drawn from too near the bottom of the main oil-storage tank, in order to prevent mechanically held bodies from being drawn into the fuel pipes and causing interruptions in the power service. Representative samples for testing are therefore important, as samples taken from near the top of a tank car or storage tank may not disclose the presence of impurities, and may show in all other respects a normal composition that is satisfactory as regards the use of the oil for fuel.

When oils are exposed to the atmosphere, complex oxidation products are formed; these, if present in small proportion only, will act like other impurities in obstructing the fine fuel passages of the fuel valves, will deposit gummy incrustations around the fuel needle, and will interfere with the satisfactory operation of the engine. The probable formation of these products should be kept in mind in storing the oil.

It is always best to provide a power installation with two small fuel filters of metal gauze in addition to the regular fuel-supply tank in the engine room, to prevent impurities from getting into the smaller fuel pipes ahead of the fuel pump or clogging the fuel nozzle. These filters should be in addition to the filters with which every good fuel pump should be equipped.

HEATING VALUE.

The heating value of an oil is important as showing the energy available for power generation. An oil otherwise desirable is the

more valuable the greater its heating value. This should be not less than 10,000 calories per kilogram, or 18,000 British thermal units per pound. These figures refer to the mean lower heating value; that is, the heat of condensation of the water resulting from the combustion of the hydrogen in the oil should be deducted from the upper or total calorific value, or heat of combustion. In comparing the heating values of different oils, they should be reduced to a common basis or standard of the water-free and ash-free substance.

WATER CONTENT.

A distinction should be made between water that is so thoroughly mixed with the oil as to form an emulsion and therefore does not separate from the oil on standing, and water that does so separate. Oils containing water of the former kind will not necessarily cause a failure of ignition, as free water is not present in sufficient proportion to prevent combustion. A higher temperature is, however, required to ignite and burn such oils, and the contained water lowers the temperature of combustion. The water of such emulsions often holds mineral salts (sodium, calcium, and magnesium chlorides and carbonates, etc.) in solution, which are crystallized out with the evaporation of the water, resulting in incrustations and wear of engine parts.

Mechanically entrained water, which will separate from the oil and accumulate in the bottom of the fuel tank, will cause failure of ignition, and if it displaces the oil in the fuel valves long enough, the engine will stop.

Water lowers the heating value of the oil, as its evaporation consumes fuel; it also lowers the temperature of the combustion space. A purchaser should not pay for water and the cost of transporting it when he is buying oil. Fuel oils should not carry more than 0.5 per cent of water, and a greater water content should be subject to a penalty.

COKE RESIDUE.

The action of coke residue has already been discussed. Oils leaving coke on distilling can not be burned in all Diesel engines. A 5 per cent coke content is probably the upper limit. Oils with an even higher content can be burned, especially in large engines, but such oils will more quickly foul the valves and the engine cylinder, and will cause increased wear. The selection of an oil will always resolve itself into the striking of a balance between the relative first cost of a cheaper oil and the relative cost of engine upkeep. There is in the United States such an abundance of fuels suitable for Diesel engines that there is no need to have recourse to fuels that produce an unduly high yield of coke.

ASPHALT CONTENT.

Asphalt is here termed matter insoluble in ethyl ether and ethyl alcohol (Holde's method). Most oils high in residues produced at a temperature higher than 400°C . are high in asphaltic substances. High asphaltum content makes an oil objectionable for use in engines with fuel valves that are closed by fuel needles. It tends to gum the needle and causes it to stick. By heating the oil sufficiently to make it more liquid, this objection is greatly removed. Such oils are as a rule highly viscous, and they have to be heated to cause the necessary fluidity.

High asphalt content points to a high content of constituents that will produce a coke residue. If the coke residue of an oil is satisfactory, its asphalt content may be disregarded as being of no importance.

PARAFFIN CONTENT.

The paraffin content of an oil is important merely in its physical effect on the oil. Oils with an appreciable paraffin content may solidify or become highly viscous at low temperatures (0° to -15°C .). Moderate heating of oils will obviate any difficulty from the sluggishness at low temperatures due to paraffin content.

SULPHUR CONTENT.

Sulphur in oil burned in the engine cylinder is converted to sulphur dioxide and may be oxidized to the trioxide. At the high temperature prevailing the percentage of sulphur usually present in fuels has no appreciable effect on cast iron. Even so high a sulphur content as 5 per cent in the fuel corresponds to less than 0.1 per cent by volume of sulphur dioxide in the gaseous products of combustion, which are replaced by a fresh volume of air every revolution in a two-stroke engine, or every other revolution in a four-stroke engine.

Mexican oils containing sulphur up to 5 per cent are successfully burned in Diesel engines; the exhaust pipes, however, should be of cast iron rather than steel, and the cooling of the gases should not be carried so far that the water vapor resulting from the combustion of the hydrogen will condense, as then the corrosive action of the sulphurous and sulphuric acids would be more destructive.

The corrosive action of sulphur dioxide gave considerable trouble in American Diesel engines of an obsolete type, in which an emulsion of lubricating oil and water was used in a closed crank case for lubricating the crank pins and the main bearings. Any of the gas that leaked past the piston was partly absorbed by the water in the crank case; the absorption was cumulative to the point of saturation and the sulphur dioxide badly corroded the crank shaft and the connecting rods.

Sulphur dioxide or trioxide may have a deleterious action on the cylinder oil, causing the formation of gummy substances and destroying the lubricating properties of the oil. The character of this action is still obscure, but usually an inferior lubricant or one that is adulterated or contains vegetable oils is indicated. Only pure mineral oils should be used for cylinder lubrication.

ACIDITY.

The fuel oils should be free from mineral acids.

ELEMENTARY COMPOSITION.

Knowledge of the elementary composition of a fuel oil is valuable as it discloses the carbon and hydrogen content of the oil. By stating the hydrogen content as so many parts of H available for every 1,000 parts of C, all oils are reduced to a common standard of comparison. Generally, a high hydrogen content points to the formation of a large proportion of oil gas that is easily ignited and burned.

VISCOSITY.

As the viscosity of an oil is a relative physical property, which changes with heat, the viscosity of fuel oils should be tested at different temperatures in the Engler viscosimeter, namely, at 20° C., 35° C., 50° C., and 75° C. Oils suitable for Diesel engines should have a viscosity of not more than 4° Engler at 75° C. If the viscosity exceeds 2.5° Engler at 20° C., the oil must be heated. Oils otherwise excellent may be too viscous at ordinary temperatures.

FLASH POINT.

The flash point of an oil is of value only as indication of the fire hazard. The flash point should be not below 60° C., and for oils used in engines with open nozzles, not below 70° C., as determined with an Abel-Pensky or a Pensky-Martens tester, corrected to a barometric pressure of 760 mm. of mercury. As flash-point determinations made in the open cup usually show results several degrees higher than those made in a closed tester, it is well to denote the instrument used in determination, as 60° CPM (60° C. in a Pensky-Martens tester).

BURNING POINT.

The burning point of an oil is the temperature at which it ignites and continues to burn in an open cup. The burning point is no criterion of the usefulness of an oil for use in a Diesel engine, save that it is a further index of the fire hazard. The nearer the burning point to the flash point, if the flash point is low, the greater is the fire hazard. A low flash point and a high burning point indicate the presence of highly volatile oils mixed with heavy oils. The burning point is 10°

to 50° C. and rarely 100° C., higher than the flash point. Extreme differences usually indicate crude oils that have not been "topped," or mixtures of volatile and of heavy residual oils. If the flash point is sufficiently high to preclude fire hazard, determination of the burning point is superfluous.

SPECIFIC GRAVITY OR DENSITY.

A density determination merely denotes in a general way whether a fuel oil is a residuum, a crude oil, or a distillate, but is otherwise of no significance in indicating the value of a fuel oil for use in Diesel engines. High viscosity is often mistaken for high density, although as regards Mexican and Californian crude oils, a dense oil is as a rule sluggish. Density is of importance when freight rates are based on volume measurement instead of absolute weight. Density values should be referred to a standard temperature of 15° C.

Petroleum fuels as herein classified comprise heavy crude oils that do not contain sufficient volatile oils to warrant the expense of "topping;" "topped" oils, from which the benzene has been removed; "gas oils," or oils distilled at relatively high temperature, the residues of which are used for road oils or converted into asphaltum; and residuums, generally classed as fuel oils, which result from oil-refining operations.

ENGINE FACTORS AFFECTING COMBUSTION.

Manufacturers of Diesel engines have constantly striven to improve and construct the engines with a view to burning a greater variety of fuels, and in particular to make possible the burning of the heavier fuel oils and residues that are produced in great abundance in oil refining.

The following factors influence the thoroughness of the combustion: Size and speed of engine; atomizer construction and atomization of the fuel; degree to which the fuel spray is mixed with the air of combustion; and the shape of the combustion space.

In small engines the provision of water-cooled surfaces for the cylinder and the cylinder head relatively large in proportion to the size of the combustion space is more effective than a similar provision in larger engines. Effective cooling is especially noticeable at low engine loads. The engine speed affects the speed of the fuel injection. In high-speed engines the fuel is injected during a period of one one-hundredths of a second, whereas in large low-speed engines the injection may last over a period of one-twentieth of a second. There are definite limits to the velocity with which a fuel can be burned; aside from being influenced by the character of the fuel, the velocity is chiefly controlled by the speed of the chemical reaction. As a part of the fuel burns with the formation of carbon dioxide and

water vapor, and possibly carbon monoxide, these gases reduce the activity of the oxygen toward the remainder of the fuel. With more time available for burning a fuel, and with equally thorough atomization, the combustion will be more thorough or, what is more important, the higher may be the boiling point of the oil.

ATOMIZER CONSTRUCTION.

The importance of thorough atomization has already been discussed. Heavy oils and residues demand higher injection-air pressure, so that the fuel will be broken into extremely small particles. The boiling-point temperatures of oils advance with an increase in pressure; further, the higher the boiling points of the oils the smaller is their vapor volume and the greater the volume of air required for their combustion. Although gasoline vapor requires for combustion about 40 times its own volume of air at the same temperature and pressure, a heavy petroleum vapor requires about 100 times its volume of air. With the increase in the volume of air, the diffusion of the fuel vapor through the air is much reduced and the obtaining of an intimate mixture of fuel and air becomes much more difficult. To bring each molecule of fuel vapor into contact with 100 or more molecules of air needed for combustion, requires a most intimate mixture of fuel and air. If this mixture is incomplete, the more volatile oil particles, which require relatively much less air, will burn first and leave particles with high boiling point incompletely burned, causing the formation of soot and oil tar.

SIZE AND SHAPE OF COMBUSTION SPACE.

A small combustion space, with highly-compressed air, greatly aids the thorough mixing of oil vapor and air. These considerations also explain why the combustion space of the cylinder should not be split up or have pockets, but have smooth walls so that the fuel particles can travel through the air more readily. For the same reason, the piston floor should be so shaped and the atomizer nozzle so constructed that the fuel spray will be spread over the entire cylinder area and, in being propelled forward by the force of the injection, air will be diffused through the air of combustion. The high temperature in a Diesel engine cylinder is another factor that results in much increased speed of chemical reaction.

SPECIAL METHODS USED IN BURNING HEAVY OILS.

Certain important adjustments, based on the foregoing considerations, are made for burning heavy oils, as follows: Increased injection-air pressure and smaller nozzle orifice to insure more complete atomization of the fuel; higher compression in the engine itself to procure higher temperature of the air of combustion; and heating of the oil

to as much as 180° F. to make it more fluid and more easily atomized. These different adjustments require considerable experience on the part of the operating engineer, who must take into account the character of the fuel used.

For oils that contain asphaltum and much oil tar at temperatures higher than 400° C. the open-nozzle type of fuel valve is more suitable, as this valve has no fuel needle, which easily becomes fouled when there is much asphaltum or tar present. In this valve the valve needle comes in contact with the injection air only—never with the fuel, which also obviates the necessity of grinding the fuel needle, as required in the closed-nozzle type of fuel valve.

Highly viscous oils, beside having to be heated, require that the engine be started and operated with gas oil until the discharge engine-cooling water is hot enough to render the viscous oil sufficiently fluid.

TWO LARGE CLASSES FOR PETROLEUM FUELS.

Suitable petroleum fuels may be divided into two classes—those that can be used in all Diesel engines, and those that can be used only in specially equipped engines. In the first class can be placed all fuels with sufficiently high flash point to minimize fire hazard, contain no asphaltum nor mechanical impurities, and have a mean lower heating value of 18,000 British thermal units per pound. Fuels of the second class have to be tried first in the engine, which often has to be especially equipped to burn them successfully

SPECIFICATIONS FOR FUEL OILS SUITABLE FOR DIESEL ENGINES.

The following specifications for fuel oils suitable for Diesel engines, published by the Rumanian Section of the International Petroleum Commission in 1912 (*Cahier des charges types pour les fournitures de produits Romains de pétrole*; Bucharest, Imprimerie de l'Etat, 1912), are of interest as they describe what oils of the first class should be.

1. The specific gravity (density) must be 0.860 to 0.895 at 15° C., to be determined with officially standardized aërometers. The density is to be corrected for each degree of temperature above or below the standard by ± 0.0007 .

2. The flash point measured by the Martens-Pensky tester must be not lower than 60° C.

3. Tests to determine the boiling-point range must be made with a 100-c. c. sample, according to standard methods in an Engler fractionation flask; at least 90 c. c. of the oil should be distilled at 350° C., and if distillation is continued, until the residue is coked, the coke should not weigh in excess of 0.5 gram.

4. Acidity. The heavy gas oil must be free of mineral acids. For testing, the use of methyl orange as applied to kerosene (lamp oil) should be used.

5. Water and mechanical impurities. The oil should be entirely clear and should not contain any suspended matter visible to the naked eye after filtering.

6. Ash content. On evaporating 50 grams of the oil to dryness in a platinum dish, no weighable quantities of ash must remain.

7. The viscosity must not exceed 2.50° Engler, measured with the Engler or the Engler-Ubbelohde viscosimeter according to standard methods.

8. Heating value, determined with the Berthelot-Mahler calorimeter, should be at least 10,000 calories per kilogram (18,000 British thermal units per pound).

It will be seen that these specifications do not allow a coke residue of even 0.5 per cent, and allow no ash content at all. It is evident that the life of an engine is materially increased, and that wear and tear and the cost of maintenance are decreased when oils meeting these specifications are used. Some oil refineries in the United States have put on the market gas oils that meet practically all of these requirements, and that are sold at a small advance in price over those of heavy fuel oils and residues. The fuel consumption of Diesel engines is so small that the increase in price is hardly reflected in the cost of the power, at least of small plants; in fact, the reduced wear and tear on the engine, the less frequent fouling of the valves, and the knowledge that the oil is of constant composition, eliminating readjustments of the fuel valves, is sure to offset the slight increase in cost. This advance over the cost of ordinary fuel oils varies from 15 to 25 cents per barrel of oil, depending on locality; on a basis of 0.5 pound of oil per effective brake horsepower-hour and 640 horsepower-hours per barrel of oil (at 320 pounds) the increased fuel cost amounts to $15/640$ to $25/640$ cent, or about 0.025 to 0.04 cent per brake horsepower-hour.

The oil specifications of a leading American builder of Diesel engines require that the oil shall have "not more than 10 per cent residue; this residue is that remaining after a sample of the oil has been reduced to approximately constant weight in a closed furnace for 120 hours at a temperature of 300° C. Gravity at 60° F. not heavier than 20° B.; not lighter than 40° B. The gravity may be heavier than 20° B., the limitation being that it shall readily flow to and be handled by the fuel pumps of the engine, to which end artificial heating may be necessary with heavy oils and in a cold climate.

"To insure the safe storage of the oil, uninterrupted operation of the engine, and minimum wear and tear, it is recommended that fuel oil also comply with the following: Flash point, between 125° and 250° F.; burning point, between 160° and 300° F.; acid, not over a trace; sulphur, not over 1.5 per cent; water, not over 0.3 per cent; ash, not over 0.01 per cent."

In marked contrast with the foregoing requirements are the fuel specifications of one of the foremost European manufacturers of Diesel engines, a large number of whose engines are operated successfully in Mexico on heavy Mexican oils. These specifications are as follows:

1. Flash point, determined by Abel-Pensky or Pensky-Martens testers, to be not lower than 70° C.
2. Viscosity not to exceed 4° Engler at 75° C.
3. Asphaltum is not objectionable as long as the viscosity of the oil does not exceed that stated.
4. The oil should be free from water, grit, loam, or similar impurities, and should contain no sand.
5. Sulphur up to 2½ per cent is not objectionable.

LIGNITE-TAR OILS.

Lignite-tar oils are the product of slow distillation at relatively low temperature of lignites and bituminous shales. They are composed chiefly of hydrocarbons of the aliphatic series, and form a valuable fuel for Diesel engines, but are chiefly of local importance in countries having lignite beds but no supply of cheap petroleum fuels.

AROMATIC HYDROCARBONS.

COAL-TAR OILS.

Coal-tar oils as used at present for Diesel engines are mixtures of the coal-tar fractions naphthalene oil and anthracene oil. From the first the naphthalene is mostly removed; and the tar oil should be as free as possible from anthracene and its homologues. These oils are mixtures of aromatic hydrocarbons. Their structural difference from the hydrocarbons of the aliphatic series has already been noted, as well as the greater difficulty of burning these oils in the Diesel engine. These oils have high self-ignition points. A cold engine can therefore not be started with them. It must first be heated by using a gas oil. Such oils will not prove satisfactory for operating an engine at fractional loads, unless an ignition oil (gas oil) is injected ahead of the tar oil to produce the heat necessary to initiate combustion; otherwise, particularly at low load, the exhaust will be very smoky.

Small engines that work with lower cylinder temperatures than those in large engines, and at higher speed, do not burn the tar oil as well as do large engines. Large engines operating continuously at full or nearly full load can burn tar oils successfully without the use of an ignition oil except for starting.

Tar oil is the principal fuel for Diesel engines in Germany, where about 1,000,000 horsepower is represented. As Germany does not

command petroleum resources of any importance and there are heavy duties imposed on petroleum fuel oils, Diesel engines were early adapted to burn the tar oils.

It is interesting to note here that the late Dr. Diesel advocated that all gas coals should be coked, so that the valuable by-products would not be wasted. The coke was then to be used to produce heat and power, furnishing a smokeless fuel. In Germany all coke is made in by-product ovens, all beehive ovens having been abolished many years ago, but Dr. Diesel wanted to see the coking extended to all coking coals mined, and to have coke instead of coal used for all metallurgical, heating, and power purposes (in locomotives, large boiler plants, and gas-producer plants). If this practice were carried into effect, Germany could develop 10,000,000 horsepower from coal-tar oils and coal tars alone, without sacrificing benzol and other valuable by-products of the coking operations; in addition there would be the possibility of producing power from the coke-oven gas, and the coke, with less waste, could be put to all the uses that bituminous coals serve.

Much of the coal made into coke in the United States is coked in beehive ovens. By the gradual change from this wasteful practice to that of coking in by-product ovens new uses for the coal tars as Diesel engine fuels could be found, without materially curtailing the production of the lighter oils, as parts of the middle and heavy oil fractions constitute the chief tar oils used in Diesel engines.

SPECIFICATIONS FOR TAR OILS.

Limiting values of the chemical and physical properties of typical coal-tar oils are given in Table 2 following, from Constam and Schläpfer.^a

TABLE 2.—*Limiting values of physical properties of typical coal-tar oil.*

Specific gravity at 15° C.....	1.006 to 1.110
Flash point, °C.....	66 to 121
Burning point, °C.....	84 to 160
Self-ignition point in platinum crucible and oxygen stream, °C.....	b 550
Boiling-point range:	
Light oil, 0° to 170° C., per cent by volume.....	0 to 12
Middle oil, 170° to 230° C., per cent by volume...	0 to 80
Heavy oil, 230° to 270° C., per cent by volume...	1 to 36
Anthracene oil, 270° to 350° C., per cent of volume.	9 to 66.5
Residue above 350° C. (pitch), per cent by volume.	1 to 34.5
Water, per cent.....	0 to 1.82
Ash, per cent.....	0 to 0.04
Heating value:	
Calories per kilogram.....	8,845 to 9,125
British thermal units per pound.....	51,921 to 16,425 *

^a Constam, E. J., and Schläpfer, P., Ueber Treiböle: Ztschr. Ver. deut. Ing., Bd. 57, 1913, p. 1621.

^b Approximately.

Heat of combustion of ash and water-free oil:

Calories per kilogram.....	9, 236 to 9, 506
British thermal units per pound.....	16, 625 to 17, 110
Carbon, per cent.....	87. 1 to 91. 4
Hydrogen, per cent.....	6 to 7. 8
Oxygen and nitrogen, per cent.....	1. 4 to 4. 9
Sulphur, per cent.....	0. 4 to 0. 9
Hydrogen available to 1,000 parts of carbon, parts.....	62 to 83
Coke, per cent.....	0. 4 to 3. 6
Free carbon and mechanical impurities, per cent.....	0 to 0. 2
Naphthalene, per cent.....	0. 8 to 10. 5

The following specifications covering tar oil suitable for Diesel engines have been adopted by the German Tar Product Syndicate and certain manufacturers of Diesel engines:

1. The tar oil must contain not more than 0.2 per cent of solid matter insoluble in xylol; the noncombustible constituents must not exceed 0.05 per cent.
2. The water content must not exceed 1 per cent.
3. The residual coke must not exceed 3 per cent.
4. At least 60 per cent by volume of the oil must be distilled below or at 300° C.
5. The mean lower heating value must be not less than 8,800 calories per kilogram (15,800 British thermal units per pound).
6. The flash point must not be below 65° C.
7. The oil must be thoroughly fluid at 15° C. If the oil is cooled to 8° C. and allowed to stand for half an hour, no solids must separate out.

In engines using coal-tar oils, all fittings coming in contact with the oil must be made of cast iron or nickel steel, as cast steel and wrought iron are severely attacked by coal-tar oil.

COAL TARS.

According to the method used in coking the coal, coal tars may be classified as horizontal-retort tars, inclined-retort tars, vertical-retort tars, chamber-retort tars, and tars made in coke-oven retorts. Of these coal tars, with a few exceptions, only those recovered in the manufacture of coal gas in vertical retorts and those recovered from the production of coke in by-product coke ovens are suitable for use in Diesel engines. In exceptional instances the coal tar from inclined retorts has been used. Coal tars vary so much in composition with the different coals and with the kind of retorts and coke ovens that a given tar must be carefully tested, chemically and in the engine, before a decision can be reached as to its usefulness as a fuel for Diesel engines. They almost invariably contain mineral constituents introduced from the retort or coking chambers, and if these constituents are of a silicious character they are particularly destructive in action on the cylinders and valves.

Tars made in horizontal retorts can not be used in Diesel engines because they contain only a small proportion of light oil and a relatively large proportion of heavy and anthracene oils, and more than half of the tar consists of pitch. The coke residue is therefore high, being often in excess of 30 per cent, with a correspondingly high free-carbon content; likewise the naphthalene content is high, as the primary distillation products are decomposed further in horizontal than in vertical retorts, owing to the uniformly higher temperatures used in horizontal retorts. These tars are black and at ordinary temperatures are highly viscous. They have the highest flash, burning, and ignition points of the gas-retort tars.

Intermediate between horizontal-retort and vertical-retort coal tars are those from inclined retorts.

Vertical-retort tars and chamber-oven tars are successfully used as fuel in Diesel engines; these tars have a high hydrogen content and therefore a higher heating value than the tars made in horizontal retorts. They are relatively high in light oils; their flash and burning points are therefore low, and they are thin fluids. They have relatively little pitch at temperatures higher than 350° C., and the free-carbon content, the coke residue (average about 6 per cent), and the naphthalene content are the lowest of all the coal tars. They are relatively low in water content and in ash.

Coke-oven tars differ from those mentioned previously in that they are "debenzolized" in the process of production, and are therefore black, thick, heavy liquids. Their specific weight, flash points, burning points, and ignition points are the highest of all the coal tars. They have high boiling points, and contain no light oil and little "middle" oil, but are relatively high in heavy and anthracene oils. They contain 50 to 65 per cent of pitch, being therefore correspondingly high in free carbon and in coke residue. The composition of these tars varies considerably, as they are produced from a great variety of coals and in different types of coke ovens.

All coal tars are relatively high in ash content, mechanical impurities, and water. Moreover, the free carbon and the coke residue foul the cylinders and the valves and these tars vary in composition and properties. Consequently they become more valuable for fuel for Diesel engines after having been subjected to a distillation or refining process. By this process the coal-tar oils already described are obtained, which are distinguished by greater purity and uniformity in composition.

In many European cities sufficient coal tar is derived from local gas works to supply the fuel requirements of the Diesel engines that provide the cities with light and power. In Switzerland particularly all power plants operated by Diesel engines, of which there is a large number, are now using gas-works coal tar or tar oils made from such tars, as the supply of petroleum fuel from the Galician fields has been cut off by the war.

WATER-GAS AND OIL-GAS TARS.

Water-gas and oil-gas tars are waste products resulting from the decomposition of gas oils from petroleum and of tars from lignite, bituminous shales, and oil shales in the manufacture of illuminating gas. The properties of these tars vary greatly according to how highly the retorts have been heated and therefore may partake more of the properties of the initial gas oil or approach those of coal tars. Their use in Diesel engines, therefore, depends entirely on their chemical and physical properties. Tars that are high in hydrogen content and low in free carbon make excellent fuels for Diesel engines, whereas those high in free carbon and naphthalene are unsuitable.

Coal-tar oils and coal tars as Diesel-engine fuel have no immediate importance in the United States, as there is an abundance of the more desirable petroleum fuels; they may, however, be of future economic importance, especially if our coal-tar production is increased with an increase of coking operations in ovens that permit the recovery of by-products.

VEGETABLE OILS.

ARACHIS OIL.

Arachis or ground-nut (peanut) oil is obtained by pressing the seeds (peanuts) of *Arachis hypogæa*, a leguminous plant, growing profusely in tropical countries. The seeds are said to contain 38 to 50 per cent of oil and the pods 4 to 5 per cent. This oil can be successfully burned in Diesel engines, and it may prove a valuable source of fuel where there are no cheaper liquid fuels, as in the interior of the colonies of some foreign countries. With that object in view its use has been developed by the French Ministry of Colonies.

PALM OIL.

Palm oil is obtained from a number of different species of palms, but chiefly from the fruit pulp of the oil palm (*Elaeis Guineensis*) growing in West Africa. The oil is a fatty substance which melts into a clear oil at about 90° F.

COMPOSITION AND PROPERTIES OF TYPICAL LIQUID FUELS.

The composition and the properties of typical liquid fuels are shown in Plate XVI. It will be noted that with an increase in density the heating value drops. There are marked differences in the viscosities of the different fuels, necessitating the heating of those that are not refined products. There is also much variation in the temperature at which the oils evaporate and in the degree of evaporation. The Galician gas oils, which are derived principally from paraffin-base oils, are cited as an example of liquid fuel extensively used in Diesel engines abroad. Only small differences between the properties of American oils and those of Mexican oils will be noted, except that

the latter have an appreciable sulphur content. The Texas fuel oils represented in Plate XVI are excellent fuels for Diesel engines, so long as the permissible ash content is not exceeded.

The California crude oils represented are unsuitable for use in Diesel engines, chiefly because of their excessive ash content and their high coke content and their decomposition at 300° C. or at a slightly higher temperature. The oils represented in the figure are not cited as being representative of California crude oils generally, but as affording a comparison between oils suitable for use in Diesel engines and oils that are not suitable for such use. The California residuum cited meets all the requirements of a desirable fuel, the ash content not being excessive, although the boiling points range fairly high and the oil has to be heated for use.

Many Mexican crude oils, as indicated, have a low heating value on account of the large water and sulphur contents; likewise, their ash content is frequently so high as to make them worthless for fuel for Diesel engines. Many contain sufficient proportions of light oil to lower their flash points and their burning points to such a degree as to make them constitute a serious fire hazard. At temperatures higher than 300° C. they decompose, leaving large proportions of coked residue.

LUBRICATING OILS FOR DIESEL ENGINES.

Two kinds of oils are used for the lubrication of Diesel engines—bearing oil and cylinder oil.

BEARING OIL.

Bearing oil is used for the lubrication of the main, the crank-pin, and the piston-pin bearings, and of the cam-shaft bearings, and for the oil baths into which the cams and the camshaft dip. A pure mineral oil will fill these requirements if it has a maximum viscosity of 7° Engler at 50° C., if it does not congeal at a temperature higher than -10° C., and if it has a flash point of 180° to 200° C. The oil should be entirely soluble in benzene, forming a clear solution without any residue; also, it should be entirely free from acids, resinous substances, and vegetable matter.

CYLINDER OIL.

Cylinder oil is used for the lubrication of the working cylinders and the air-compressor cylinders. It should be stable at the relatively high temperatures of the piston and the cylinders of a Diesel engine, and should maintain the desired lubricating properties at these temperatures; it should not vaporize readily, nor should it decompose at a temperature lower than 300° C.

BUREAU OF MINES

		GALICIAN GAS OILS	OKLAHOMA TEXAS GA
1	LOWER HEATING VALUE, B.T.U. PER POUND		
2	SPECIFIC GRAVITY AT 15 °C.		
3	VISCOSITY "ENGLER"		
4	CONGELING POINT	°C. +5 to -15	-1 to -
5	EVAPORIZATION CURVE, CUMULATIVE PER CENT		
6	FLASH POINT, °C.		
7	BURNING POINT, °C.		
8	ELEMENTARY ANALYSIS	C% 86.2 H% 12.65 NO% 1.75	C% 86.2 H% 12.6 NO% 1.0
9	ASH	PER CENT NONE	NONE
10	SULPHUR	PER CENT 0.2-0.8	0.2
11	WATER	PER CENT NONE	NONE
12	FREE CARBON	PER CENT	
13	COKE RESIDUE	PER CENT 0.2	0.3
14	BARBITHELE	PER CENT	



DESIRABLE PROPERTIES OF CYLINDER OIL.

A suitable cylinder oil should have a viscosity of 9° to 10° Engler at 50° C., and should flow at -5° C. Its flash point should not be below 240° C. It should be a pure mineral oil free from asphaltum and pitch and resinous substances, and should be completely soluble in benzine, leaving a clear solution without any residue. A piece of sheet brass covered for 24 hours with the oil, at 100° C., should remain perfectly bright and show no action by acids, nor should it be gummed by any substance. A thin layer of the oil spread on the sheet of brass should contain no solid particles perceptible to the sense of touch.

Even the best oils will become partly oxidized in the compressor and engine cylinders and form complex pitch-like substances and insoluble compounds, which absorb fine metal particles from the compressor cylinder or dust that is carried into the cylinder with the air. With good oils it takes many months to form such accretions, and their harmful effect can be entirely avoided by a periodical inspection and cleaning of the cylinders.

Cylinder oils should not be decomposed in the working cylinders nor the air-compressor cylinders, as decomposition permits the more volatile part of the oil to burn and form carbon, which covers the piston, causes the piston rings to stick, destroys the lubricating effect of the oil, and results in the scouring of the pistons and cylinders, and sometimes in the seizure of the pistons.

Users of Diesel engines can make no more serious mistake than to purchase a low-priced cylinder oil in an attempt at economy. A high-grade cylinder oil, sparingly used, will prove most economical as regards both the quantity of oil consumed, and the cost of engine upkeep. Oil analyses alone are insufficient to govern the selection of a desirable oil. The best way of testing the suitability of an oil is to note its performance in a Diesel-engine cylinder. The best guide in the purchase of lubricating oils is the recommendation of the engine builders, as their recommendation is usually based on the results of protracted tests of different lubricating oils.

IMPORTANT ADVANTAGES OF DIESEL ENGINES.**LOW PRICE OF FUEL.**

The high commercial value of the Diesel engine lies in its unsurpassed fuel economy and in its ability to burn low-priced liquid fuels with high boiling points. This great fuel economy is not confined merely to medium or large-sized units, for the smallest engine has nearly the same fuel economy as the largest. In this respect Diesel engines differ from steam engines, which become economical only in large sizes.

SELF-CONTAINED PRIME MOVER.

The Diesel engine possesses the further great advantage of being a self-contained prime mover, its only auxiliary being the air compressor, which in all high-grade engines is driven direct from the engine crank shaft, and is mounted on the engine frame. If directly connected to a direct-current generator or an alternator with excitor, it comprises with the switchboard equipment a complete electric generating station. Its space requirements are therefore small, and it can be placed in basements of buildings and hotels.

MINIMUM FUEL-STORAGE SPACE REQUIRED.

The small fuel requirements of the engine reduce to a minimum the the space necessary for fuel storage, and as the engine uses liquid fuels, these can be piped by gravity from tank cars to the storage tanks below track level. A 10,000-gallon tank car will supply enough fuel to drive a 100-horsepower engine at full load continuously, day and night, for more than two months. Noncondensing steam engines of the same size consume 4 to 6 pounds of coal or $2\frac{1}{2}$ to 3 pounds of oil per effective horsepower-hour, or the equivalent of 340 to 500 tons of coal, or 200 to 250 tons of oil as compared with the 40 tons of oil used by the Diesel engine. The saving in storage space is evident.

The labor of unloading and storing coal, firing boilers, and removing the ashes, which may readily amount to one-quarter of the coal fired, is entirely eliminated in Diesel-engine plants. The absence of coal, ash, and smoke makes for cleanliness. The Diesel engine is noiseless and its exhaust is colorless and odorless, so that its operation meets all hygienic requirements.

FULL POWER QUICKLY AVAILABLE.

The engine is in readiness for instant operation, and can take on full load from no load as quickly as the operator can make the different shifts, a matter of only two or three minutes. With the stopping of the engine no further fuel is consumed. In this respect the Diesel engine differs from heat engines in which the working fluid is generated outside of the engine—in boilers for steam engines and in gas producers for suction-gas engines. The Diesel engine has no stand-by losses. Boiler or gas-producer losses in firing up, losses up the stack, losses in incompletely burned and clinkered coal, radiation and condensation losses, losses due to inefficient firing of boilers or their insufficient upkeep, or, in suction gas plants, losses due to inefficient producer operation, to afterburning caused by incorrect setting of the valves or timing of the ignition—these add considerably to the increased

heat consumption of steam and gas engines at fractional loads. In this respect the Diesel engine differs from other prime movers in that its over-all fuel economy is not dependent on the efficient operation of auxiliaries, or even on the skill of the operators, as its operation is largely an automatic thermodynamic process in the engine itself.

RELIABILITY.

With a correctly designed and efficiently built engine this economy will be maintained undiminished during the life of the engine, provided the engine receives proper care.

The claim of greater reliability for the steam engine is frequently made, because an engine supplied with steam from a boiler will continue to run even if piston and glands leak, or the valves are improperly set for operating economy, and no matter how wastefully the boiler may be operated; whereas the Diesel engine has to generate its own driving fluid in the engine cylinder, and if any part of the engine fails the engine has to stop. However, the supposedly greater reliability of the steam engine is obtained at the expense of a greatly increased fuel consumption.

The Diesel engine has been developed to such a degree of perfection during the past decade that carefully built engines can be considered as being fully as reliable as steam equipment.

No attempt will here be made to outline comparative costs of generating power in the large central stations producing thousands of kilowatts. These unusually large stations, however, are relatively rare, the average size of all central stations in the United States not greatly exceeding 500 kilowatts; and the power demands of most industrial plants are met by small and medium sized plants.

As is outlined subsequently, many industrial plants can generate their own power at less cost than they can purchase current from large central stations.

The simplicity and compactness of Diesel-engine plants is well illustrated by figures 43 and 44 (pp. 100-101), showing complete vertical and horizontal engines.

COMPARATIVE EFFICIENCIES OF DIFFERENT PRIME MOVERS.

Table 3 following shows the heat consumption and the thermal efficiencies of different types of prime movers at continuous full load. The high figures of fuel consumption refer to small plants, the low figures to larger plants; they represent good economy in well-operated plants equipped with modern high-grade machinery.

TABLE 3.—*Heat consumption and thermal efficiencies of different types of prime movers at continuous full load.*

Type of prime mover.	Heat consumption per brake horsepower-hour.	Over-all thermal efficiency.	Superiority of Diesel engine. ^a
	<i>B. t. u.</i>		
Noncondensing steam engine ^b	40,000-28,000	6.30- 9.1	5.6-3.6
Condensing steam engine using superheated steam ^b	28,000-16,500	9.1-15.4	3.6-2.3
Locomotive engine with superheated steam and reheater, condensing ^b	17,000-15,200	14.9-16.7	2.4-2.1
Steam turbine, superheated steam, 200 to 2,000 horsepower ^b	24,000-15,500	10.6-16.2	3.2-2.2
Steam turbine, superheated steam, 2,000 to 10,000 horsepower ^b	15,500-14,000	16.2-18.1	2.2-1.9
Gas engine without producer.....	10,400- 9,300	24.4-27.5	1.33-1.25
Suction gas engine ^c	14,000-11,200	18.1-22.7	1.95-1.55
Diesel engine.....	8,000- 7,200	32-35.3	

^a Figures in this column are to be used as factors with which to multiply values in preceding column.^b Figures include boiler losses.^c Figures include producer losses.

COMPARATIVE COST DATA.

COST OF OPERATION.

Table 4 following presents fuel-cost data covering the operation of the prime movers represented in Table 3:

TABLE 4.—*Comparative operating costs of different types of prime movers.*

Type of prime mover.	Kind of fuel.	Average cost per 1,000,000 B. t. u.	Cost of 1,000,000 B. t. u., effective work.	Heat cost per one effective horsepower-hour.
		<i>Cents.</i>	<i>Cents.</i>	<i>Cents.</i>
Noncondensing steam engine ^a	Coal.....	12	191-132	0.48-0.34
Condensing steam engine using superheated steam ^a	do.....	12	132- 78	.34-.20
Locomotive engine with superheated steam and reheater, condensing ^a	do.....	12	81- 72	.21-.15
Steam turbine, superheated steam, 200 to 2,000 horsepower ^a	Coal and anthracite.....	11	113- 74	.29-.19
Steam turbine, superheated steam, 2,000 to 10,000 horsepower ^a	do.....	12	104- 68	.27-.17
Gas engine without producer.....	(Natural gas, coke-oven gas, or blast-furnace gas.....	11	74- 67	.19-.17
Suction gas engine ^b	Anthracite.....	11	68- 61	.17-.155
Diesel engine.....	Petroleum.....	15-18	62- 55	.16-.14
		7	27	.07
		11	61- 49	.16-.14
		15-18	56- 43	.14-.11

^a Figures include boiler losses.^b Figures include producer losses.

COST OF FUEL.

Cost data covering the various types of fuel used in the prime movers represented in Tables 3 and 4 are presented in Table 5 following:

TABLE 5.—*Cost of various types of fuel.*

Kind of fuel.	Price of fuel.	Heating value per pound.	Absolute heat cost per 1,000,000 B. t. u.	Average heat cost per 1,000,000 B. t. u.
		<i>B. t. u.</i>	<i>Cents.</i>	<i>Cents.</i>
Lignite.....	\$1.00 to \$2.50, ton of 2,000 pounds.....	5,000- 9,000	10 -14	12
Bituminous coal.....	2.00 to 4.00, ton of 2,000 pounds.....	11,000-14,200	9 -14	12
Anthracite.....	2.50 to 4.00, ton of 2,000 pounds.....	14,500	8.6-13.8	11
Fuel oil (petroleum).....	.75 to 2.25, barrel.....	18,000-19,000	12.5-37.5	15-18
Natural gas.....	.10 to .75, 1,000 cubic feet.....	900- 1,000	11-75	15
Blast-furnace gas.....	.05 to .01, 1,000 cubic feet.....	90	5.5-11	7
Coke-oven gas.....	.02 to .05, 1,000 cubic feet.....	450	4.4-11	7

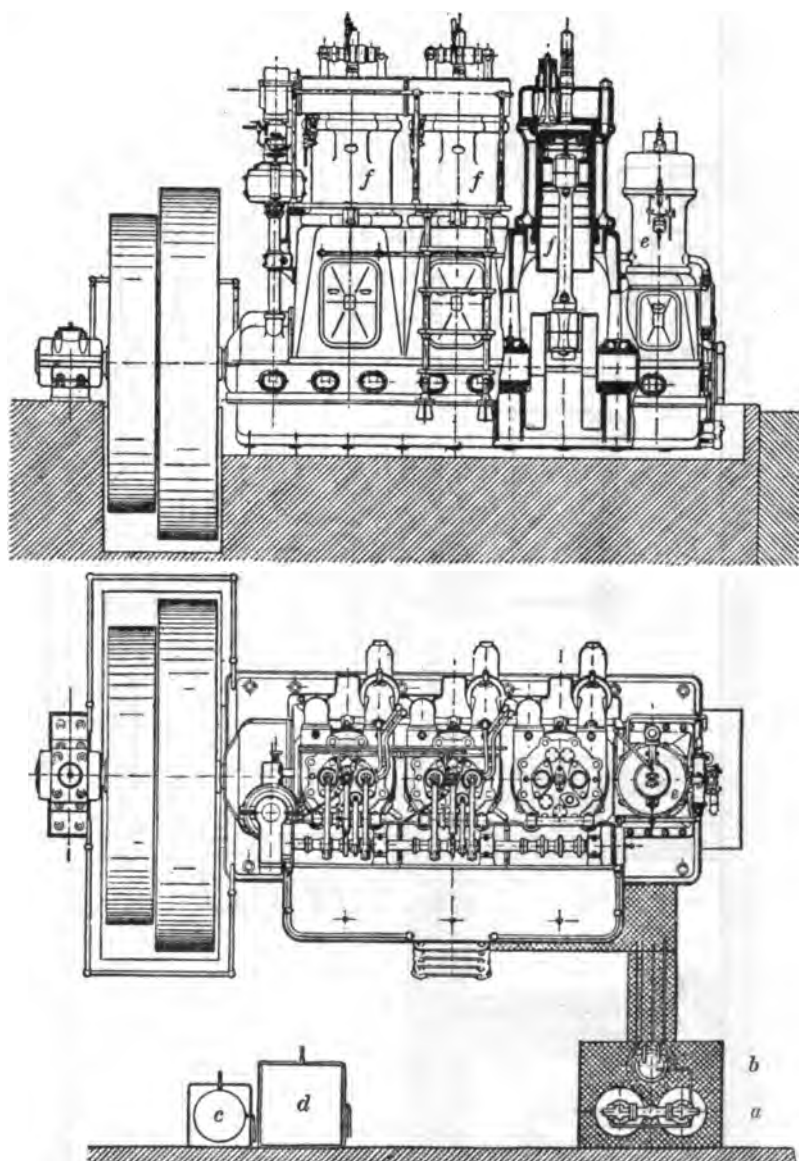


FIGURE 42.—Plan and elevation of vertical three-cylinder Diesel engine with all auxiliaries: *a*, air receiver; *b*, injection air bottle; *c*, ignition-oil tank; *d*, fuel-oil tank; *e*, air compressor; *f*, working cylinders.

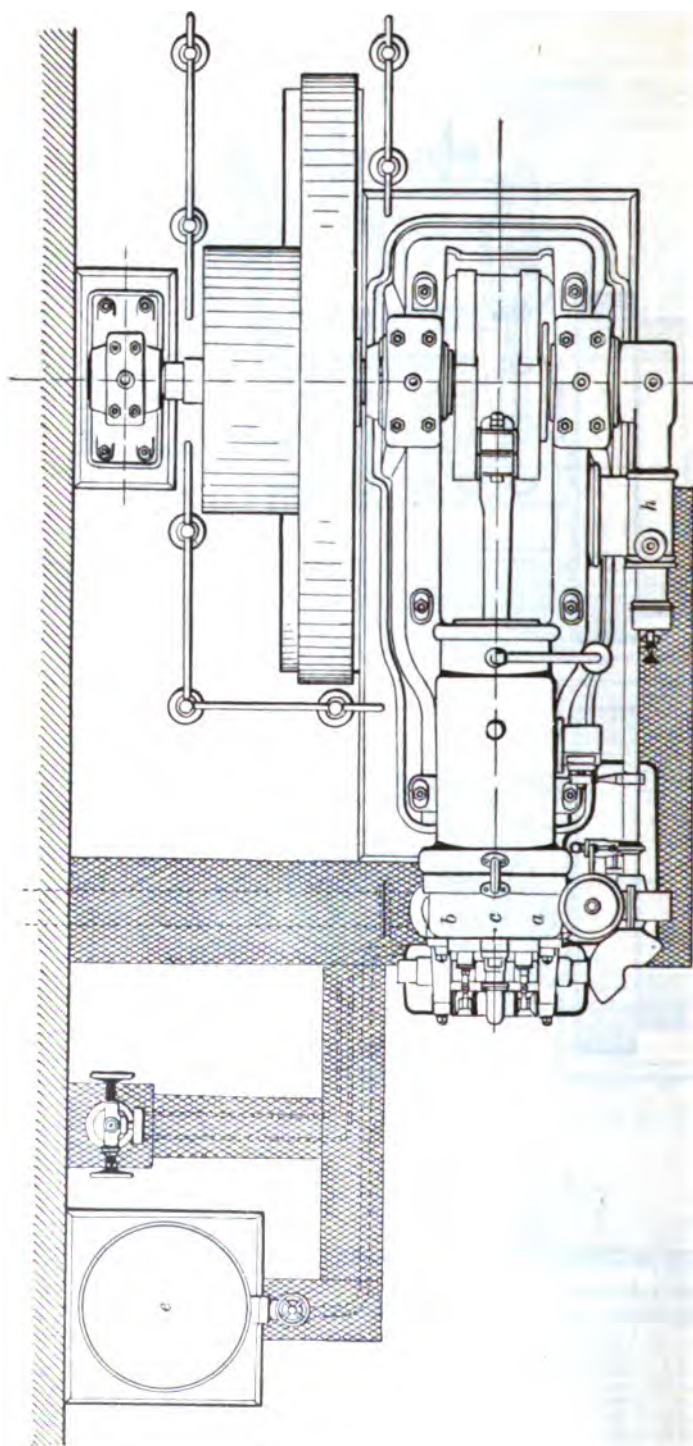


FIGURE 43.—Plan of horizontal Diesel engine with all auxiliaries. *a*, air valve; *b*, exhaust valve; *c*, fuel valve; *d*, starting valve; *e*, receiver for starting air; *f*, receiver for injection air; *g*, fuel tank; *h*, air compression.

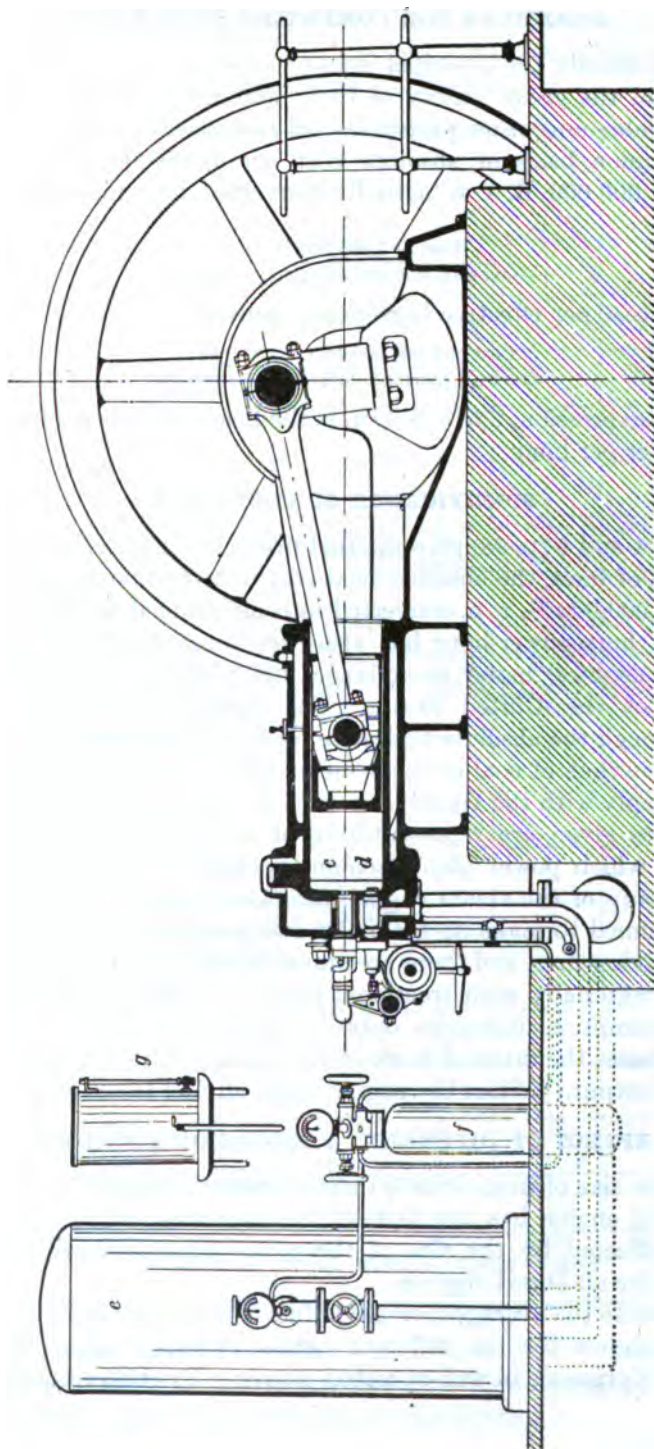


FIGURE 44.—Elevation of engine shown in figure 43.

FORMULAS FOR COMPUTING FUEL COST.

Fuel is usually the principal item of expense in generating power. Because of the great variety of fuels, and their differing greatly in heating value and other properties, all fuel prices should be reduced to a common basis of absolute cost per 1,000,000 B. t. u. The cost of 1,000,000 B. t. u. actually converted into mechanical work is then :

$$\frac{\text{Cost of 1,000,000 B. t. u.}}{\text{Over-all thermal efficiency of plant.}}$$

The fuel cost per effective horsepower-hour is

$$\frac{\text{Cost of 1,000,000 B. t. u.} \times 2,545.}{\text{Over-all thermal efficiency} \times 1,000,000.}$$

In this expression, 2,545 B. t. u. is the heat equivalent of 1 English horsepower per hour.

SIGNIFICANCE OF COST DATA.

Tables 4 and 5 (p. 98) give the fuel prices and heat values of different kinds of fuels, the absolute heat cost of 1,000,000 B. t. u. and the cost of 1,000,000 B. t. u. converted into mechanical work. The heat cost of 1 horsepower-hour has also been computed for the different kinds of engines, based on average fuel prices in widely different sections of the United States. The figures mentioned indicate that although the absolute fuel price for fuel oil may be several times higher than that of coal in the same market, power may be generated more cheaply with the Diesel than with the steam engine. However, the figures given are representative of continuous full-load conditions, at which power plants seldom operate. At fractional loads, the economy of the steam engine decreases; there is also additional fuel consumed to make up for boiler and producer losses when firing up or banking fires, and there are stand-by and radiation losses which increase materially with fractional loads. In the Diesel engine the fuel consumption increases only in proportion as the fractional load increases the internal work of the engine, with a decrease in its effective output. When the engine stops all fuel losses stop.

EFFECT OF DIFFERENT FACTORS ON ECONOMY.

How the size of units affects their economy is shown by figure 45. The Diesel engine has the highest fuel economy, which, moreover, is least affected by the size of the unit. The economy is nearly constant for all Diesel engines.

In figure 46 the average increase in fuel consumption with fractional loads is shown for the different prime movers, based on actual engine performance in well-operated plants. To obtain the economy

of steam plants at fractional loads, the full-load economy for a given size of plant can be taken from figure 45 and then the percentage increase computed for that size from figure 46.

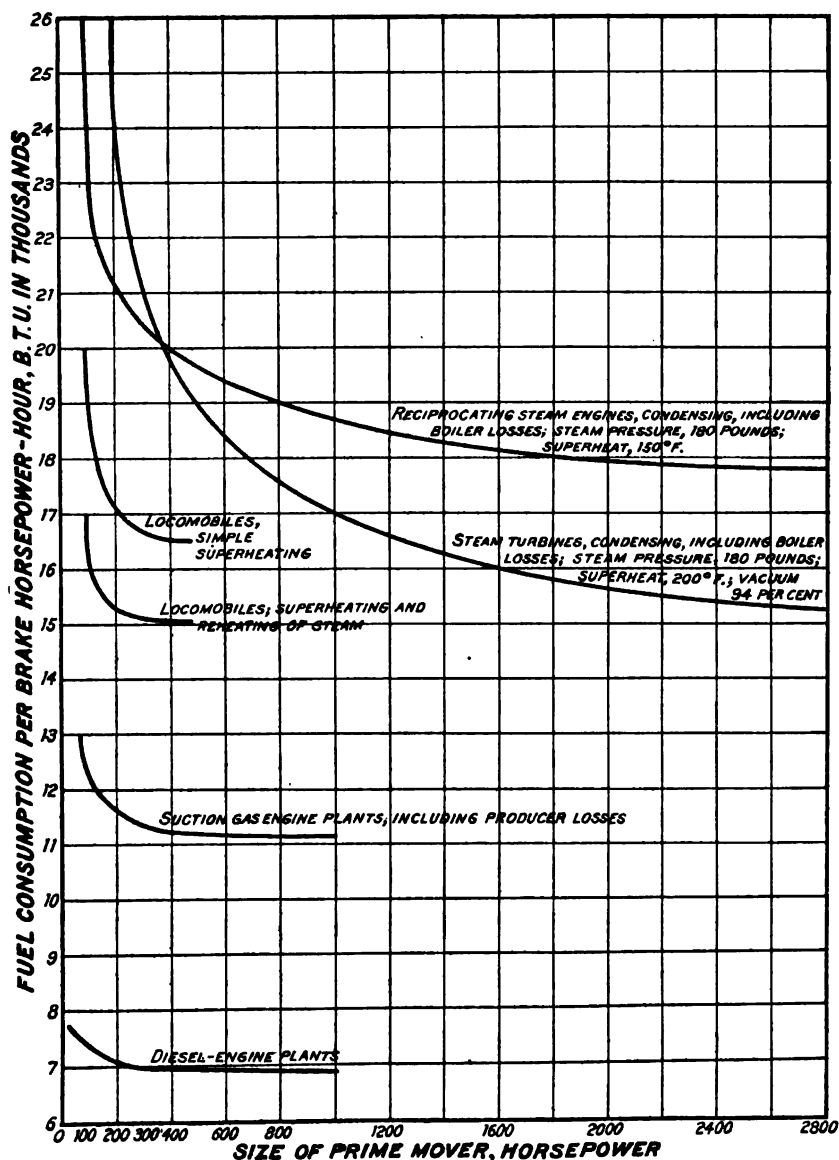


FIGURE 45.—Influence of size on efficiency of different prime movers at continuous full load.

With inexperienced labor and insufficient care of machinery, the fuel consumption of the steam and the suction gas power plants may be still higher than is shown in the figures.

From the data that has been presented it is evident that small steam plants can not compete with Diesel engines in the cost of generating power. On the basis of results of the actual operation of steam and of

modern Diesel-engine plants, allowance being made for the special factors favoring the Diesel engine, such as small space requirements, constant readiness for immediate operation, cleanliness, and absence of danger from possible boiler explosions, steam plants up to 1,000 horsepower, using coal costing 10 cents per 1,000,000 B. t. u. can not effectively compete with Diesel-engine plants even with fuel costing 30 cents per 1,000,000 B. t. u.

ADVANTAGES OF LOCOMOBILES.

Among the types of steam engines considered, the locomobiles occupy a place apart, on account of their phenomenal economy for their small size. The locomobile is a self-contained power plant, consisting of a boiler, a superheater, and a compound condensing engine, a condenser, a soot blower, and a feed-water heater. It is usually equipped also with a reheater in which the steam is reheated before it is admitted to the low-pressure cylinder. These units are

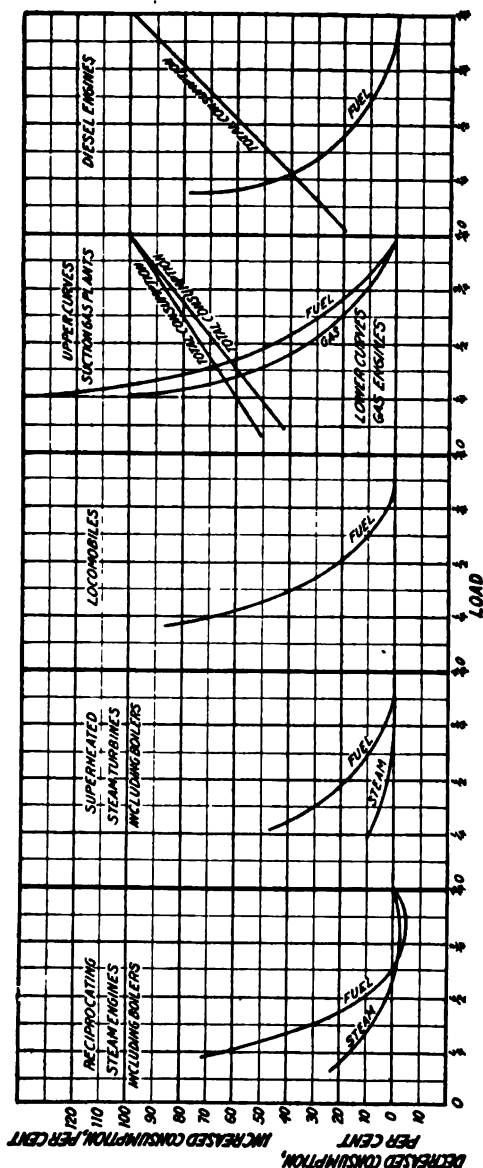


FIGURE 46.—Average increases in fuel consumption by different prime movers at fractional loads.

widely used in Germany and other parts of Europe, and in recent years an American concern has begun to manufacture them.

In actual everyday operation, however, these units do not meet the performance guaranties established by test. A recent investiga-

tion of the performance of 12 locomobile plants and 15 Diesel-engine installations in different parts of Germany disclosed the fact that during one year's operation with the changing load conditions common

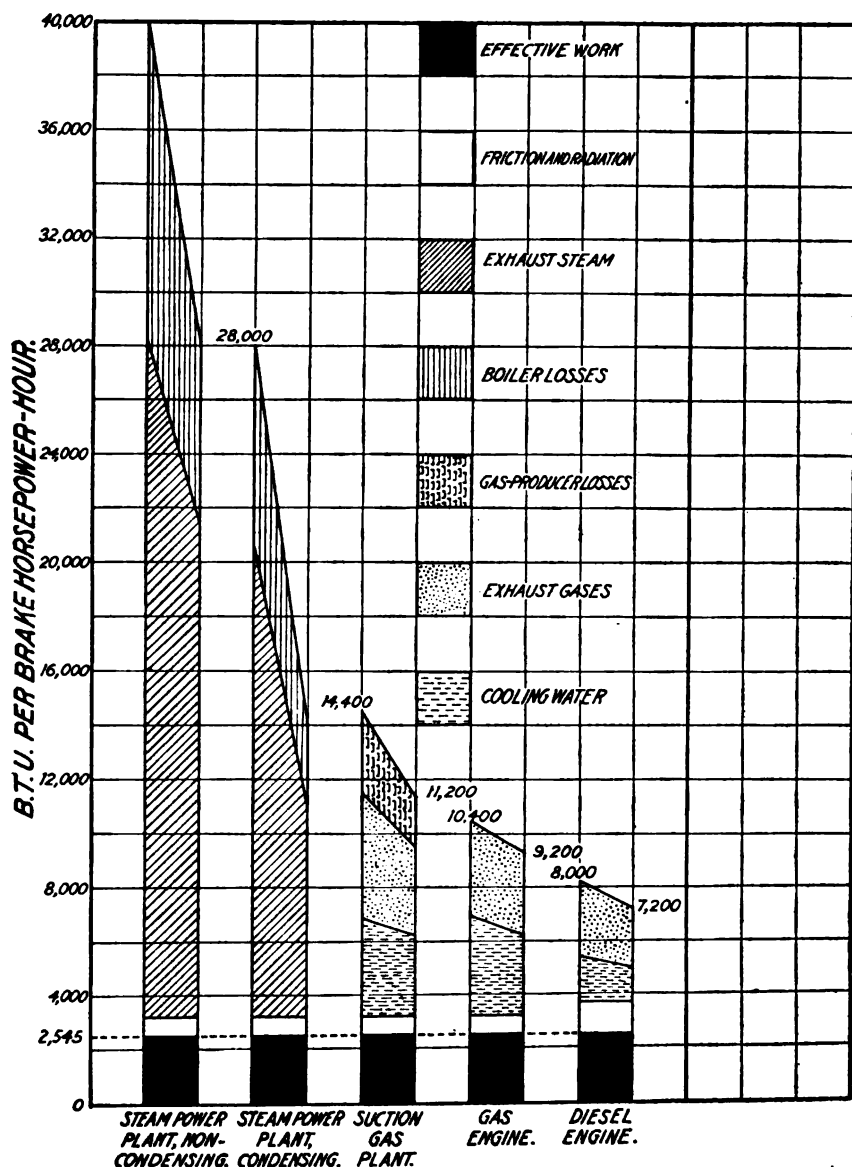


FIGURE 47.—Heat balances of different kinds of engines.

to most central stations the actual fuel consumption of the locomobile plants averaged 102 per cent higher than the guaranteed performance, whereas the consumption of the Diesel-engine plants was only 14 per cent higher than the performance guaranteed.

LIMITATIONS OF SUCTION GAS PLANTS.

The locomobile units, on account of their economy, may be used advantageously in sections of the country where fuel oil is not obtainable or where the price is high as compared with the price of coal. Similarly, a suction gas plant may be in order where cheap anthracite or coke is to be had and fuel oils are so high in price as to offset the manifold advantages and the superiority of the Diesel engine over suction gas plants. With the use of bituminous coal, the operating difficulties of suction gas engine plants increase, and not all bituminous coals can be used successfully in suction gas producers. Success depends greatly on the experience of the operator in making a uniform gas of the desired composition. Further, the valve setting and the timing of ignition must be adjusted to the composition of the gas. On account of a lack of experienced labor to operate such plants, disinclination on the part of users to pay advanced wages for such experience, or, possibly on account of deficient design of the producer or the gas engine or both, many such plants have failed, and they have not been as widely adapted as their relatively high fuel economy would lead one to expect. With the present development of the Diesel engine, a further decline in the use of suction gas engine plants may follow. Suction gas engine plants cost fully as much as Diesel-engine plants per horsepower of capacity; they take up more room for equipment and for fuel storage, and depend on the producer for their gas supply. Their fuel consumption under average operating conditions, even in well-conducted plants, is seldom better than $1\frac{1}{4}$ to $1\frac{1}{2}$ pounds per horsepower, compared with about one-half pound of fuel oil for Diesel-engine plants.

CONDITIONS MAKING OTHER PLANTS DESIRABLE.

When a supply of natural gas, coke-oven gas, or blast-furnace gas is available, the gas engine is generally the logical selection. If the heat price of these gases as well as the load factor of the station is very low and the load is highly fluctuating, large steam-turbine plants are the correct economical selection at present, as they cost least per kilowatt of capacity. The reduction in capital cost overbalances any fuel saving possible with gas engines, this saving being relatively small because the absolute fuel consumption for the limited kilowatt-hour output of such stations is small.

The selection of the steam engine in small units when power is to be transmitted electrically to motors is justified only when the exhaust steam can be used either for heating buildings or in industrial processes.

The steam turbine has the advantage of lower installation cost, smaller space and foundation requirements, and oil-free steam. The steam consumption of small turbine units is, however, materially

higher than that of steam engines; their use is therefore warranted only when the amount of heat to be supplied by the exhaust steam is very large.

Figure 47 shows the comparative value of the steam engine for heating purposes. This figure shows the distribution of the amount of heat wasted per horsepower-hour after the amount required for conversion into mechanical work (2,546 B. t. u.) has been deducted. In condensing steam engines, 12,000 B. t. u. per horsepower-hour and in noncondensing steam engines 20,000 to 24,000 B. t. u. per horsepower-hour are available in the exhaust steam for heating purposes, whereas in gas and Diesel engines only about 2,000 and 1,200 B. t. u. per horsepower-hour can be abstracted from the waste gases. If the waste gases are utilized for heating, the steam engine generates power cheaply, the fuel chargeable to power production being reduced to about 8,000 to 16,000 B. t. u. per horsepower-hour.

The cooling water and the waste gases from medium and large sized Diesel engines can be used for heating when only a limited supply of hot water is needed. The safety of the engine precludes heating of the cooling water to a temperature higher than 160° F. In large engines, to avoid excessive heating of cylinders and cylinder heads, the cooling water should not be discharged at a temperature higher than 125° F. If the exhaust gases are used to heat this water (of which about 5 gallons per horsepower-hour is needed), still further, in specially constructed heaters or boilers, the efficiency of the engine drops off rapidly with an increase in the temperature to which the water is heated, as does the volume of water so heated. Five gallons per horsepower-hour may be thus heated from 125° to 170° F., and only 2½ gallons if heated to 212° F. The volume of water that can be heated is roughly reduced in proportion to a reduction in the indicated power of the engine. The Diesel engine is therefore used principally for generating power.

FACTORS AFFECTING COST OF GENERATING POWER.

As regards the generation of power with heat engines the following factors influence the cost:

1. The fuel consumption of the generating units.
2. The B. t. u. price.
3. The size of generating units and of the station.
4. The station-load factor or the ratio of yearly output in kilowatt-hours to the maximum possible yearly output in kilowatt-hours.
5. Capital or fixed charges, comprising interest on the capital invested in the power plant and amortization.
6. Operating expenses comprising cost of labor, lubrication, and maintenance.

The influence of these factors on the cost of generating power with small Diesel engines is shown in figures 48 and 49. One hundred horsepower and 200-horsepower engines which drive industrial machinery direct from belt have been selected. The engines are loaded at an average of about 80 per cent of their rated capacity, a condition commonly found in industrial plants. Labor is figured at \$3 per man per 8-hour day. The installation cost, including building space for housing the engine, is taken at \$100 per horsepower of installed capacity for the

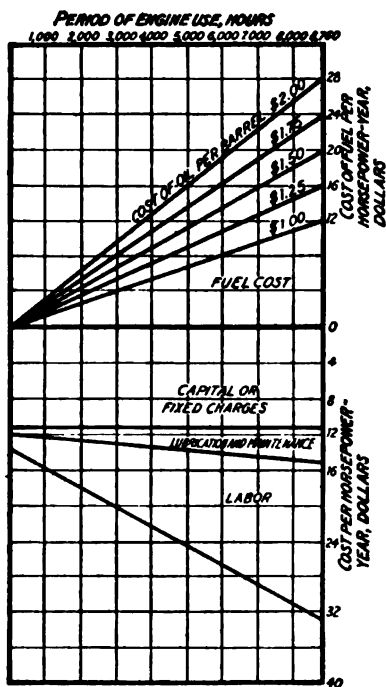


FIGURE 48.—Comparative costs of different items in the operation of a 100-horsepower Diesel engine at a mean 80 per cent load.

100-horsepower engine and at \$90 for the 200-horsepower engine. All costs are referred to a unit of 1 horsepower operating one year of 8,760 hours. The total costs of operating the engine for shorter periods are also shown. The figures show that capital charges are constant, regardless of load; that the fuel cost is proportionate to the operating time; and that the labor and the maintenance charges are nearly proportionate to the operating time.

The cost graph of the 200-horsepower engine (fig. 49) shows the influence of size on the reduction of capital and of labor charges. There is no difference in the fuel consumption of the two engines, as

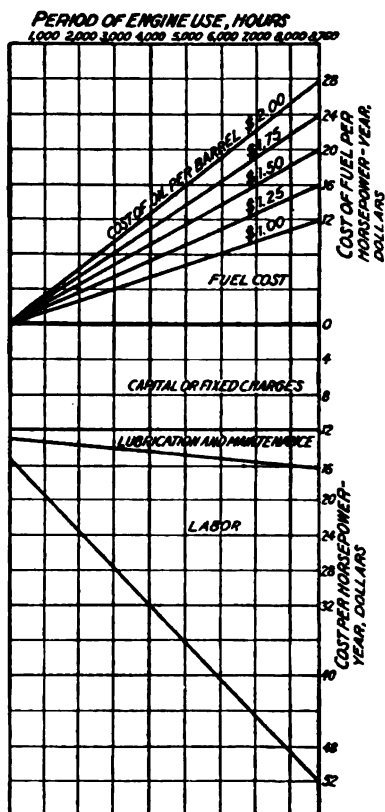


FIGURE 49.—Comparative costs of different items in the operation of a 200-horsepower Diesel engine operating at a mean 80 per cent load.

the fuel economy of both is the same. Figure 50 shows for the 100-horsepower engine the operating costs for different engine-use periods reduced to costs per horsepower-hour corresponding to these use periods or load factors. Figure 51 shows similar data for the 200-horsepower engine. The influence of a reduced number of operating

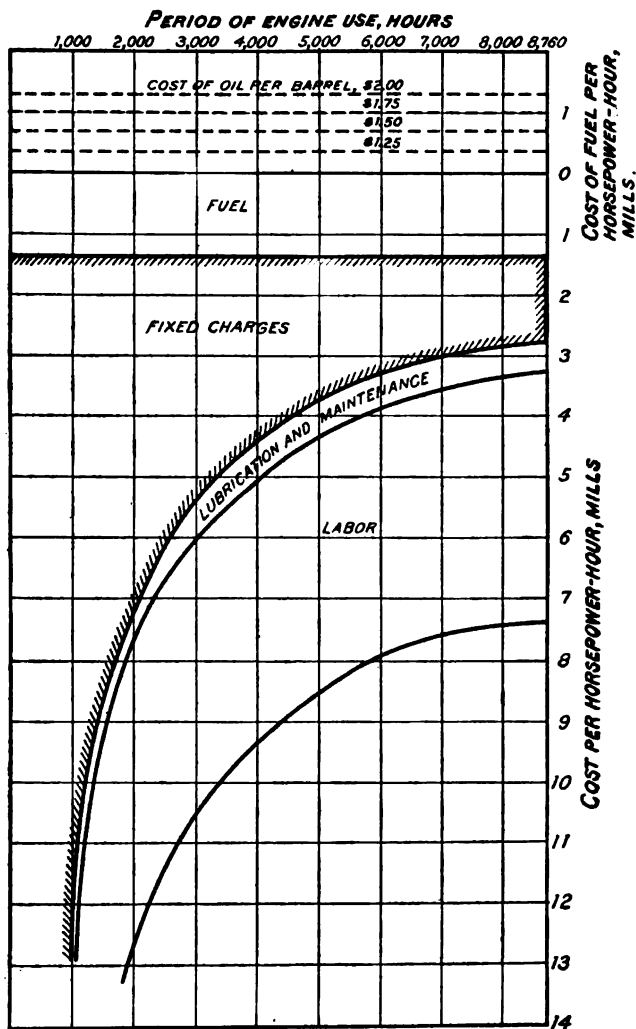


FIGURE 50.—Operating costs for different engine-use periods reduced to corresponding horsepower-hour costs, 100-horsepower Diesel engine.

hours, which is synonymous with a reduced output of horsepower-hours, is at once apparent. The fixed charges increase the unit power cost rapidly at low-use factors. For the fuel cost a basic price of \$1 per barrel of 320 pounds has been used. To what extent higher fuel prices increase the power cost is indicated by broken lines

above the base line. The cost increment proportionate to the higher fuel price would have to be added to the horsepower-hour cost shown in the graph below the base line (in which the fuel cost at \$1 per barrel is already included) for any use factor. As this cost increment is the same for both engines, figure 49 includes only the fuel cost based on oil at \$1 per barrel.

Figure 53 shows the influence of the size of Diesel-engine equipments for generating electric power on the cost of producing the power. The capital charges are based on the costs per unit of installation as given in figure 52. To obtain the best operating economy and flexibility, each station is considered as being composed

of at least three units, each with a capacity of one-third of the aggregate station capacity. With an increase or a decrease in load during different periods of the day one or more units can be thrown into or out of action.

Figure 54 shows the cost of a kilowatt-hour for different station-load factors and for stations of different sizes, the data being based on the kilowatt-year costs in figure 53. The fixed charges are represented by full lines marked KW. F. C., and the costs for labor, lubrication, and maintenance are represented by broken lines marked KW. L. M. The fuel costs have not been added, as these are practically constant at 2 to 2½ mills per kilowatt-hour for a load factor of 100 to 25 per cent, with fuel oil costing \$1 per barrel of 320 pounds. For oil of a higher price the proportionate higher fuel cost can easily be calculated.

Combined charges for capital, labor, lubrication, and maintenance can be read direct for any load factor from the broken-line curves. The line representing kilowatt-hour costs for a 6,000-kilowatt station has been cross-hatched to make it distinct. As the size of a station materially influences the cost of power production, different sized stations are indicated by separate curves to make clear the influence of station size and station factor on the power production cost.

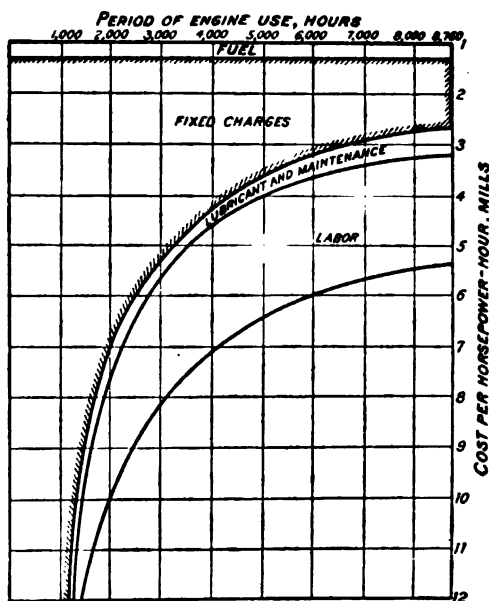


FIGURE 51.—Operating costs for different engine-use periods reduced to corresponding horsepower-hour costs, 200-horsepower Diesel engine.

Figure 55 presents a comparative analysis of the factors controlling the cost of generating power in medium-sized (9,000-kilowatt) central stations using either Diesel engines or steam turbines. Figure 56 shows the comparative efficiencies of the two types of prime movers. For the steam turbine highly favorable operating conditions have been taken, namely a steam pressure of 180 pounds per square inch; a steam temperature of 600° F.; a temperature of 60° F. at the inlet for the circulating water for condensing, and a vacuum of 96 per cent.

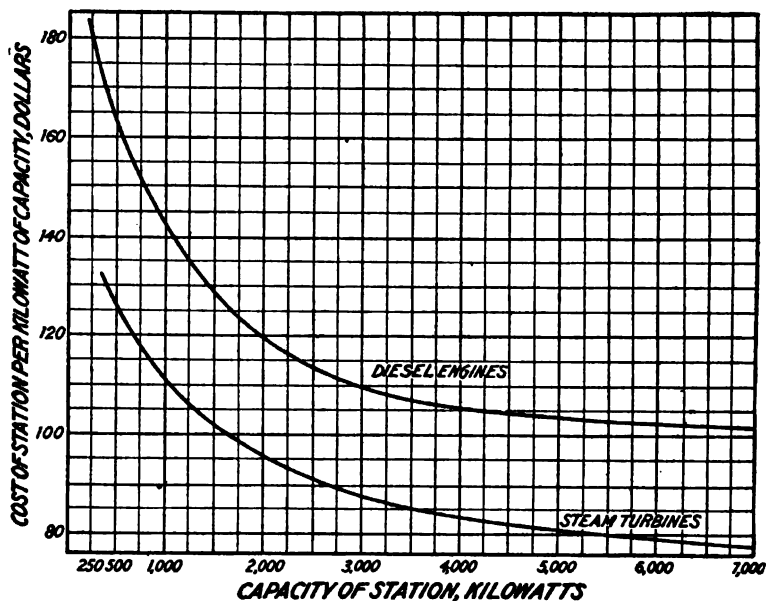


FIGURE 52.—Comparative costs per unit of Diesel-engine equipment and of steam-turbine equipment for generating electricity.

Of the three variables, steam pressure, steam temperature and cooling-water temperature, the last two affect the economy most, as is shown in figure 57, from which the increased or decreased fuel consumption can be figured for other operating conditions than those assumed. The percentage of fuel increase (above the base line marked "cooling-water temperature") and the percentage of fuel decrease (below the base line marked "cooling-water temperature") to be added to or to be deducted from the performance consumption used as a starting point can be readily ascertained for different sets of operating conditions.

Figure 55, showing the total yearly capital, lubricating, maintenance, and fuel charges for different station factors, indicates that the fuel price and the station factor (relation between average load and rated capacity) determine which type of plant is the more economical for a given set of conditions.

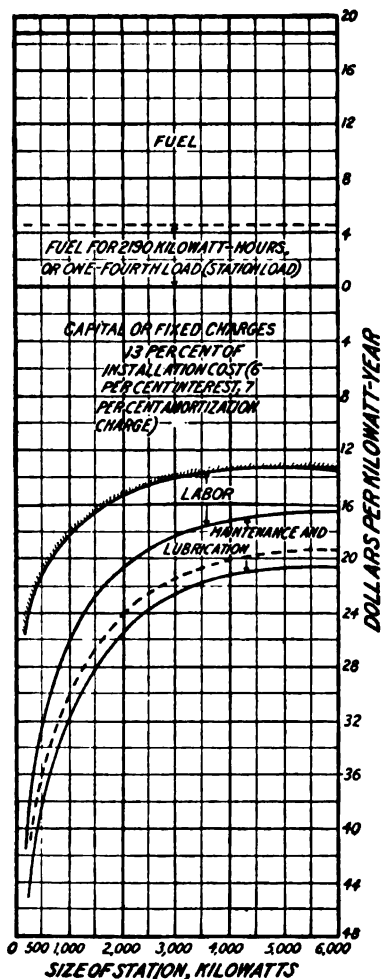


FIGURE 53.—Curves showing cost of kilowatt-year (8,760 kilowatt-hours) at full load, as influenced by station capacity.

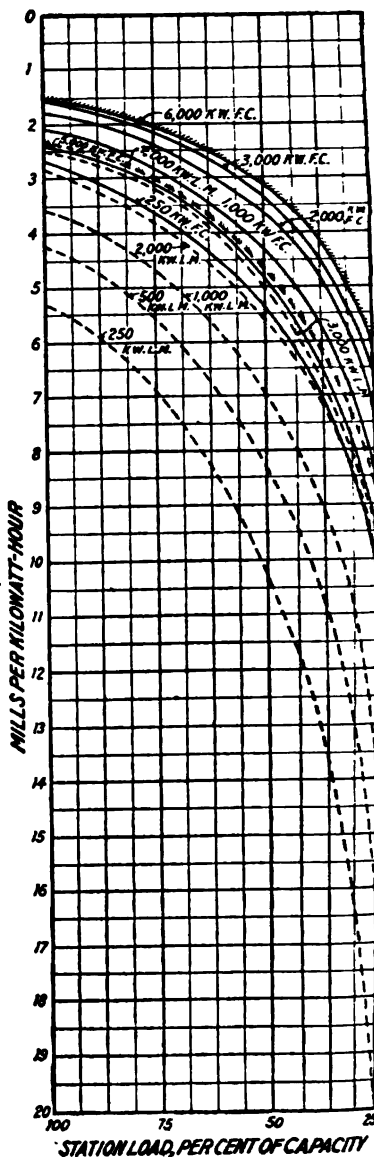


FIGURE 54.—Curves showing cost of kilowatt-hour in mills at different station load factors. Fuel cost per kilowatt-hour is 2 to $\frac{3}{4}$ mills, which must be added to costs shown.

SUMMARY OF COMPARATIVE ADVANTAGES.

Following is a statement of the comparative advantages and disadvantages of Diesel engines and of steam turbines for generating power:

Summary of comparative advantages and disadvantages of Diesel engines and of steam turbines for generating power.

ADVANTAGES OF STEAM-TURBINE PLANTS.

1. Unlimited size of generating units.
2. High rotative speed, reducing size of generating units, and cheapening turbine and alternator construction.
3. Low installation cost.
4. High overloading capacity.
5. Ability to use under the boiler all classes of fuels (solid, liquid, or gaseous), including the cheaper varieties of inferior-grade coals.

DISADVANTAGES OF STEAM-TURBINE PLANTS.

1. The fuel economy is greatly dependent on the skill of the operating force and the degree of thoroughness with which the installation is maintained at operating efficiency.
2. Dangers and drawbacks of boiler operation.
3. Boiler losses generally and fuel losses in firing up and in banking boilers and in ash and clinkers.
4. Exacting requirements relating to circulating water for condensing the steam, one of the deciding factors in the commercial success of a steam plant.

ADVANTAGES OF DIESEL-ENGINE PLANTS.

1. Very high thermal efficiency and lower absolute heat rate, independent of the size of unit.
2. Absence of all boilers, smoke, and soot.
3. Constant readiness for operation.
4. No fuel losses in firing up and in unconsumed coal in ash as in steam plants.
5. Small volume of cooling water required.
6. Use of cheap liquid fuels, such as petroleum residues, coal-tar oils, and certain coal tars.

DISADVANTAGES OF DIESEL-ENGINE PLANTS.

1. Limited size of units (at present, 4,000 horsepower).
2. Massive foundations necessary.
3. High consumption of lubricating oil.
4. High installation cost.

The fuel costs are figured for different B. t. u. prices, ranging from 10 to 20 cents per 1,000,000 B. t. u. Those represented in the right graph of figure 55 are based on the use of coal under the boilers. As higher boiler efficiencies are obtained with fuel oil than with coal, and there is also less labor involved in caring for boilers, allowance for a decrease in cost for a steam-turbine plant using fuel oil should be made comparing such a plant with a Diesel-engine plant on the basis of the data represented in the figure. Thus, the next lower price than that shown for a given fuel cost can be taken. For instance, if fuel oil costs 16 cents per 1,000,000 B. t. u., take the cost given for oil at

14 cents as representing the fuel cost for the turbine plant and compare it with the 14-cent price for the Diesel-engine plant.

CONSIDERATIONS GOVERNING CHOICE OF PRIME MOVERS.

The data represented in figure 53 indicate that the selection of either type of prime movers should be governed by the following economic considerations:

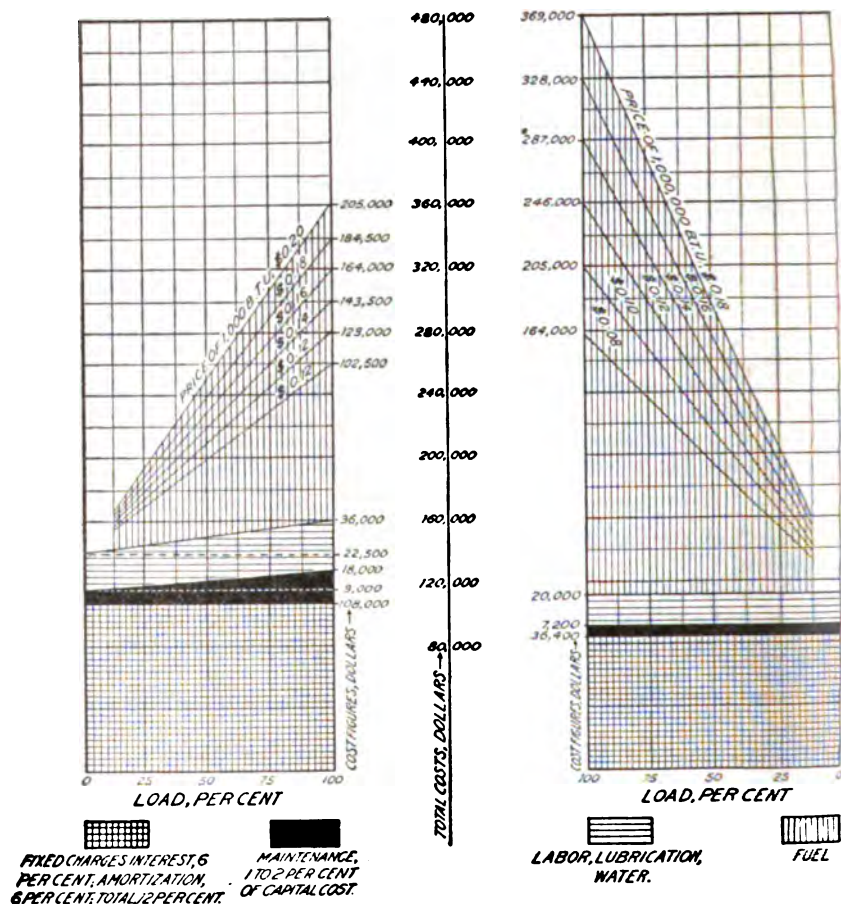


FIGURE 55.—Comparative costs of operation of Diesel-engine and of steam-turbine installation for generating power. Left graph represents four 3,500-horsepower Diesel engines in a plant costing complete, including buildings, \$900,000; right graph represents three 3,000-kilowatt steam turbines in a plant costing complete, including buildings, \$720,000.

1. Fuel is of chief influence on the total cost of power when both the price per 1,000,000 B. t. u. and the load factor are high. These conditions favor the use of the Diesel engine.

2. Interest and amortization charges are of chief influence on the total power cost with low B. t. u. price and low load factor, particularly as applied to stand-by plants, which are operated only occasion-

ally or to take care of recurring peak loads. The installation cost of such plants must be kept as low as possible so as to avoid heavy capital charges distributable over a relatively small output of kilowatt-hours for the station. Such conditions favor the steam turbine.

3. An exception to consideration 2 is when the constant readiness of the Diesel engine makes it preferable, installation cost being of secondary importance. Here the cost of keeping boilers under steam continuously would have to be balanced against the difference of interest charges on steam-turbine plants as compared with those on Diesel-engine plants.

4. The prime movers most logical for power plants situated at the source of the fuel, either in the oil field or at the coal mine, and therefore enjoying the advantages of a cheap fuel supply, are those costing least to install—that is, steam turbines. The fuel expenditures will be comparatively less than interest and redemption charges, provided that a satisfactory supply of cooling water is available for condensing.

5. In many instances combination plants using Diesel engines for supplying the continuous and nearly constant main load, and steam turbines for furnishing periodically occurring peaks by the use of high-duty boilers with large water and steam spaces, capable of being forced when necessary, will prove most profitable. The periodical peaks may be taken care of by a turbine floating on the line and operating in parallel with the Diesel engines that supply the main load and operate constantly at or near full load.

6. Up to capacities of 1,000 horsepower, steam turbines can compete with Diesel engines only for special uses, such as supplying exhaust steam for heating. For larger plants, of 1,000 to 10,000 kilowatt capacity, careful analysis must be made of the relative advantages of Diesel engines and of turbines, a knowledge of the load factor, the fuel prices, and the water supply being necessary.

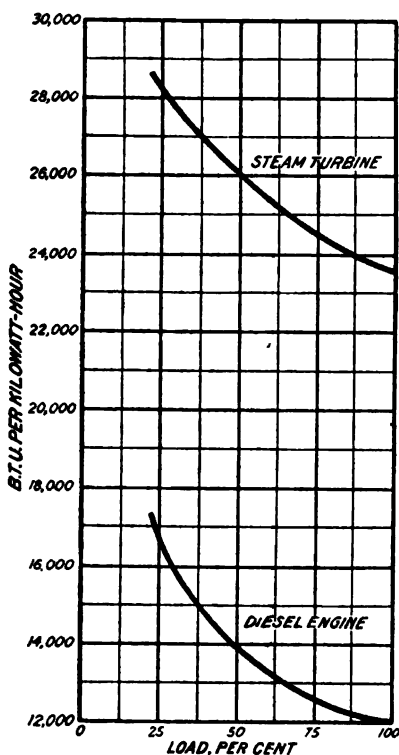


FIGURE 56.—Comparative operating efficiencies of Diesel engines and of steam turbines. Steam-turbine equipment to generate 3,000 kilowatts; figures include boiler losses and operation of auxiliaries; steam pressure, 180 pounds per square inch; steam temperature, 600° F.; temperature, at inlet, of circulating-water for condensing, 60° F.; vacuum, 96 per cent. Single-acting, two-stroke, vertical, Diesel-engine equipment to generate 3,500 horsepower; 125 revolutions per minute; 3,000-kilowatt generator.

For power plants with a capacity larger than 10,000 kilowatts, comprising units of 6,000 kilowatts or more, steam turbines are preferable, unless a combination of high load factor, high fuel cost, and unfavorable water supply favors the Diesel engine. This combination is not frequently encountered.

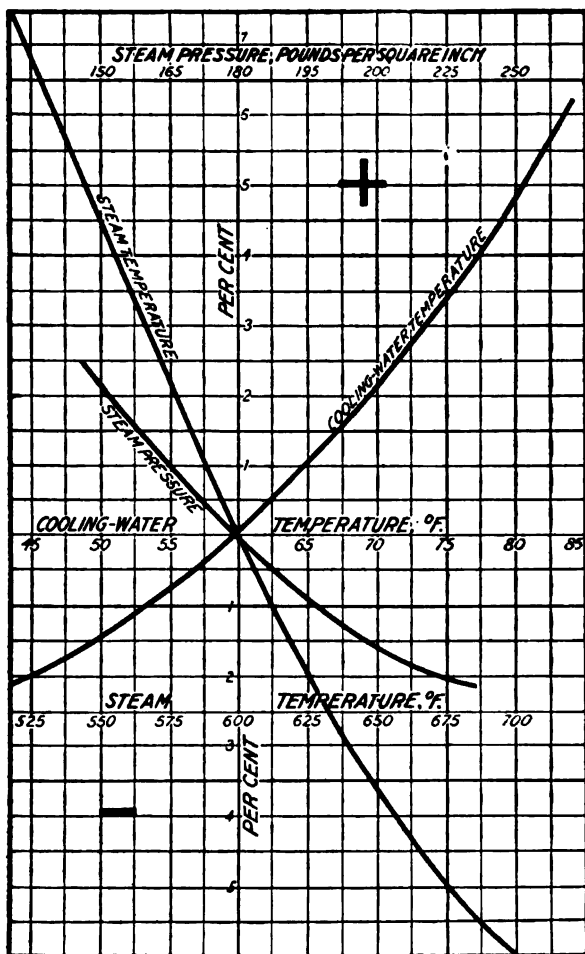


FIGURE 57.—Influence of operating conditions on performance.

SPECIAL ADAPTATIONS OF DIESEL ENGINES.

Mention has been made of combination plants in which steam turbines and Diesel engines are used. Great economies are possible with such plants. As steam turbines are flexible, can be greatly overloaded, and are relatively cheap per kilowatt of capacity, they can be used to furnish peak loads of short duration. Such loads occur in almost every central station in the early night hours, when the maximum demand on the station may be several times the

average demand. Although the steam unit consumes considerably more fuel, the absolute total fuel consumption of the steam equipment for the kilowatt-hour output at the peak is comparatively small, as the Diesel engine would be running at full-load capacity during the maximum current output of the station. During the hours in which the steam turbine is not in service, the exhaust gases and the discharged cooling water from the Diesel engine would be used for keeping the boiler heated and under steam. The use of the exhaust gases for this purpose is, of course, applicable only to the moderately large station.

Many small steam plants now in operation can make appreciable fuel economies by continuing the use of the present steam equipment for supplying the peak load and by installing sufficient Diesel-engine equipment to provide the average load. The most economical size and number of Diesel-engine units in relation to existing steam equipment can be determined only on the basis of the average daily load for a period of a year, and each power problem has to be considered separately. A practical example of the savings possible is of interest. A fluctuating yearly load of 1,250,000 kilowatt-hours was supplied by a 500-kilowatt steam turbine, operating on the condensing system, with an average fuel consumption of nearly $6\frac{1}{2}$ pounds of coal per kilowatt-hour. By the installation of a 200-kilowatt Diesel engine unit, at a cost of \$30,000, a yearly fuel saving of \$11,000 could have been realized. In this instance the coal was high priced, costing nearly 20 cents per 1,000,000 B. t. u. Such a degree of saving is possible in steam plants all over the United States, the fuel saving paying for the Diesel-engine equipment in three to four years without necessitating any increase in labor for operating such mixed plants.

Analysis of the different factors controlling the cost of generating power with Diesel engines shows clearly that, in common with other types of power plants, this cost diminishes with an increase in the size of the plant. The reduction in power-generating costs is due, first, to decreased interest and redemption charges for larger plants, these charges being in proportion to the capital outlay for the plant, and, second, to a lessened labor charge. A possible conclusion is that large central stations must produce the power cheaper than smaller independent and isolated engine units. However, the influence of the station factor on the power cost was also made evident. Furthermore, the figures given for cost of generating power were for current delivered at the station switchboard. They did not include the cost of distributing the current, which is often a more expensive operation, when the interest and redemption charges on transmission and distributing systems and the cost of power losses are taken into account.

The cost of transmission and distributing systems and the expense of their maintenance depend on such a variety of conditions that no generally applicable cost data can be given, but each system must be designed for the particular conditions to be met.

There are, however, large fluctuations in the capital cost (including interest charges and amortization) of different systems per kilowatt of capacity, and, what is more important, per kilowatt-hour of delivered energy. This cost is greater for a thinly settled distributing area and, if there are few power consumers in the area, the transmission line must be comparatively longer for each kilowatt of used output, and the power consumption will be more irregular. The power losses of such systems are also considerable. These factors all combine to make hydraulic power by no means as universally cheap as many people believe. Fixed charges resulting from the high capital cost of some of the large hydraulic-power and electric-transmission systems, and the unavoidable power losses of such systems, add so much to the total cost of delivered energy that Diesel-engine plants can often generate power more cheaply than can hydroelectric systems. The advantage of the Diesel engine is especially apparent if it can be applied direct without the double transformation of energy—mechanical into electrical and this back to mechanical. The Diesel engine is also preferable for central stations supplying current within a limited area, as the necessity of high-tension long-distance transmission is obviated. It is well recognized that large power companies frequently are obliged to sell current to large consumers at cost, or even below cost, in order to create a demand for more power. The increase in power output insures a more favorable load factor, but the numerous small consumers have to pay high prices to make up for the losses in profit on the power supplied to the large consumers.

The great economy of the Diesel engine makes even moderate-sized central stations for small cities and towns and isolated power plants feasible and renders possible the generation of such cheap power that the large central stations with overland-transmission systems can compete with them only in densely settled areas.

Most industrial plants that operate day and night, and therefore have an exceptionally high load factor, can generate their power with Diesel engines at such a cost, including all operating and fixed charges (12 to 15 per cent), that they can fix a profitable price that no central station can equal even in powers as small as 100 horsepower. Even with a 50 per cent load factor, such an engine will, as a rule, generate power cheaper than it can be purchased.

Of late years, especially in Europe, much has been done to develop the oil engine in small powers, to take the place of the electric motor. There is a great demand for such small-powered engines, particularly in agriculture. There is also a large field for small engines of

this type for tractors and motor trucks. The great weight of the Diesel engine and the first high cost per horsepower of capacity was against their adoption in the earlier stages of their development. Likewise, the necessity for thorough vaporization of the fuel was not so well understood, and the use of a compressor for the fuel injection made the cost of the small units excessive, not to mention the great difficulties of building so small an air compressor that would be reliable with the high pressures used. The clearance space and the valves would be so small that they would be fouled by a minute quantity of dirt and oily deposit.

For small powers a highly successful type of engine has been developed, in which the measured fuel charge is deposited during the suction stroke in a small perforated vaporizer extending slightly into the cylinder from the cylinder head. During the compression stroke a part of the charge is vaporized, and it is ignited at the termination of the stroke; the resulting explosion sweeps the remainder of the less readily vaporized fuel through the perforations and into the cylinder, in which it is burned completely. Gas oils are successfully burned in these engines, with a fuel consumption of 0.5 to 0.6 pound per horsepower-hour.

A further development lies in building small two-stroke engines of this type, to permit a lower weight and lower manufacturing costs. A fuel consumption of 0.6 to 0.7 pound per horsepower-hour is rather common in these engines, which range from 5 horsepower per cylinder up. What their development will mean to farmers and users of motor trucks in a time of rising gasoline prices may be realized from a comparison of fuel prices. Gas oil may be had at \$1 to \$1.50 per barrel, or 2 to 3 cents per gallon, whereas distillates and gasolines cost 9 to 25 cents per gallon. The fuel saving is therefore nearly proportional to the fuel prices, as the thermal efficiency of these small oil engines is fully as good as that of distillate and gasoline engines. Standardized engines of the type designed for mass production will reduce the manufacturing cost to such a figure as to make their adoption general throughout the country.

EXAMPLES OF SUCCESSFUL USE OF DIESEL ENGINES.

A few practical examples of uses to which Diesel engines are adapted will show what performance may be expected. The Diesel engine is the preeminent prime mover for continuous day-and-night duty at a high load, the fuel saving in such use being large in comparison with the consumption of any other heat engine, and the operating and fixed charges for the Diesel engine so used being the lowest for the unit of power.

PUMPING PLANTS.

The table following shows the amount of work performed, in foot-pounds per 1 B. t. u. of heat consumed, in lifting water moderate

heights with different types of pumping equipment. These performances relate to city water works and irrigation pumping plants.

Data regarding amount of work performed by different types of pumping equipment.

Type of pumping plant.	Work performed in lifting water.	Over-all efficiency.
	<i>Foot-pounds per 1 B. t. u.</i>	<i>Per cent.</i>
Steam; good operating conditions.....	71.5	7
Steam; best operating conditions.....	77.8	10
Steam; superheated steam used.....	93.4	12
Suction gas engine; good operating conditions.....	116.7	15
Suction gas engine; special conditions.....	147.8	19
Humphrey gas pump; good operating conditions.....	156	20.1
Humphrey gas pump; special conditions.....	171.2	22
Diesel engine; good operating conditions.....	225.6	29

A noteworthy Diesel-engine pumping station is that at the Gladstone Docks, Liverpool. There are five units, each consisting of a single-stage centrifugal pump coupled direct to a 1,000-brake horsepower Carels Diesel engine. The engines have a two-stroke cycle, with four cylinders 510 mm. in diameter by 660 mm. stroke, and deliver 1,000 brake horsepower at 180 revolutions per minute. They can be overloaded 10 per cent. The pumps, of the Worthington type, are designed to deliver 9,200 cubic feet of water per minute against a head of 47 feet at 180 revolutions per minute, but are capable of working against a maximum head of 61 feet.

Diesel engines are also successfully used for driving pumps of petroleum pipe lines when the oil has the desired viscosity at ordinary temperatures.

ICE MANUFACTURE.

Developments in the manufacture of ice have made possible the utilization of Diesel engines for driving the ammonia compressors, with a noteworthy fuel saving and reduction in the cost of manufacture. Combination ice, refrigeration, and electric-light plants are now frequently installed. The off-peak engine capacity is used for operating the ice-making machinery, with highly satisfactory results. Four to 6 gallons of fuel oil per ton of ice is the usual consumption in ice plants in which Diesel engines are used. In a plant producing 100 tons of ice in 24 hours and with fuel costing $2\frac{1}{2}$ cents a gallon, a ton of ice can be produced for 45 cents, including all operating labor, maintenance, and supply charges, but not fixed charges (interest and amortization). If the power were purchased at 1 cent per kilowatt-hour the cost would be 85 cents per ton of ice.

FLOUR MILLS.

A number of flour mills use Diesel engines. A common performance is the milling of one barrel of flour with a fuel-oil consumption of 0.5 to 0.6 gallon, for all power, including that used in elevating the

wheat and corn. With oil costing 2 cents a gallon, the fuel cost per barrel of flour for all power is 1 to 1.2 cents, and with 3-cent oil the cost is 1.5 to 1.8 cents per barrel of flour.

CENTRAL STATIONS.

The factors controlling the cost of generating electric current have already been discussed. Following are data showing unit power-production costs for a modern central station in Texas. The station is equipped with three 500-horsepower Diesel engines direct connected to 437-k. v. a., 2,300-volt, 3-phase, 60-cycle alternators. The cost figures are for a period of four months.

Station output, kilowatt-hours	1, 565, 000
Rating of plant, kilowatts	1, 050
Station factor, per cent.	51
Total fuel oil used, gallons.....	149, 072
Weight of oil per kilowatt-hour of output, pounds.....	0. 672
Heat consumed per kilowatt-hour of output, B. t. u.	13. 100
Cost of plant per kilowatt of capacity, dollars.....	153
Labor cost, average man-day, dollars	2. 80
Man-days per 1,000 kilowatt-hours of output.....	. 46
Cost of fuel oil per gallon, cents.....	3. 25
Cost of lubricating oil per gallon, cents.....	30
Production costs, per kilowatt-hour, mills:	
All labor	1. 44
Fuel oil.....	3. 07
Water.....	. 09
Lubricants and waste.....	. 04
Miscellaneous supplies and expense.....	. 10
Maintenance of engines.....	. 04
Maintenance of buildings.....	. 05
All other maintenance.....	. 15
Total.....	4. 98
Interest and amortization, 12 per cent, 51 per cent station factor.....	4. 80
Total cost.....	9. 78

LOCOMOTIVES.

The Diesel engine has been applied to railroad traction in two forms; directly in a Diesel locomotive, and indirectly in an electric motor car with a generator driven by a Diesel engine.

A 1,000-horsepower locomotive has been built by Sulzer Bros., of Winterthur, Switzerland, in which power to the drivers is transmitted direct from a reversible Diesel engine with a two-stroke cycle. For starting, compressed air, furnished by a compressor operated by a separate Diesel engine is used. The locomotive made satisfactory test runs in hauling freight and passenger trains, but no detailed performance data are as yet available.

Variable-speed Diesel engines coupled to generators and built into passenger railroad cars have been used in several instances in Europe with great success. The fuel consumption varied from 0.8 pound to 1.2 pounds per train-mile, with a train weight (two cars) of 30 to 66 tons (2,000 pounds). The average fuel consumption amounted to 2 to 2.75 pounds per 100 ton-miles. On a basis of 1 kilowatt-hour being generated with a fuel consumption of 0.7 pound of oil, this is equal, in round numbers, to 3 to 4 kilowatt-hours per 100 ton-miles, or 30 to 40 watt-hours per ton-mile, a remarkably good performance.

SHIP PROPULSION.

An increasing number of ships are being fitted with Diesel engines. Lloyd's Register for 1914 listed 27 ocean-going ships equipped with Diesel engines with a total of about 40,000 shaft horsepower. At least 20 more ships equipped with Diesel engines were then building. There are probably more than 400 ships, including river craft not listed, fitted with Diesel engines, not counting submarines so fitted. Up to 1914, 6,000 indicated horsepower, equal to about 4,600 to 4,800 shaft horsepower, in six single-acting cylinders and in one engine was probably the upper limit in size, although double-acting engines of 12,000 horsepower in three cylinders for use on ships are being tested on the European continent.

Before directly reversible marine Diesel engines had been developed, reversing and slow-speed changes were accomplished by the Del Proposto system, first used on the Russian ship *Sarmat* in 1904, and subsequently installed in many Russian river boats and ships of the Russian Navy. The engine was used for driving the ship ahead, but for reversing was coupled to a generator and a motor was used. The engine speed could be reduced from 240 to 72 revolutions per minute.

Later, Hesselmann developed a directly reversible four-cycle marine engine, which has been used with great success in many ocean-going ships. Late development favors the engine with a two-stroke cycle for marine purposes, as it can be built materially lighter per horsepower, is mechanically better adapted for reversing, and occupies much less space than other types. Compressed air is used for reversing both types of engines.

The advantages of ships equipped with Diesel engines lie not merely in the greater cruising radius obtainable with a given quantity of fuel, but in the increased cargo space made available, as the liquid fuel can be stored between the double bottoms of the ships, in dead space otherwise filled with ballast water.

The importance of Diesel engines for propelling submarines is commonly known. The latest types of submarines with their greatly increased field of action and cruising radius are almost wholly due to recent improvements in the marine Diesel engine.

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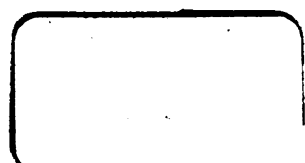
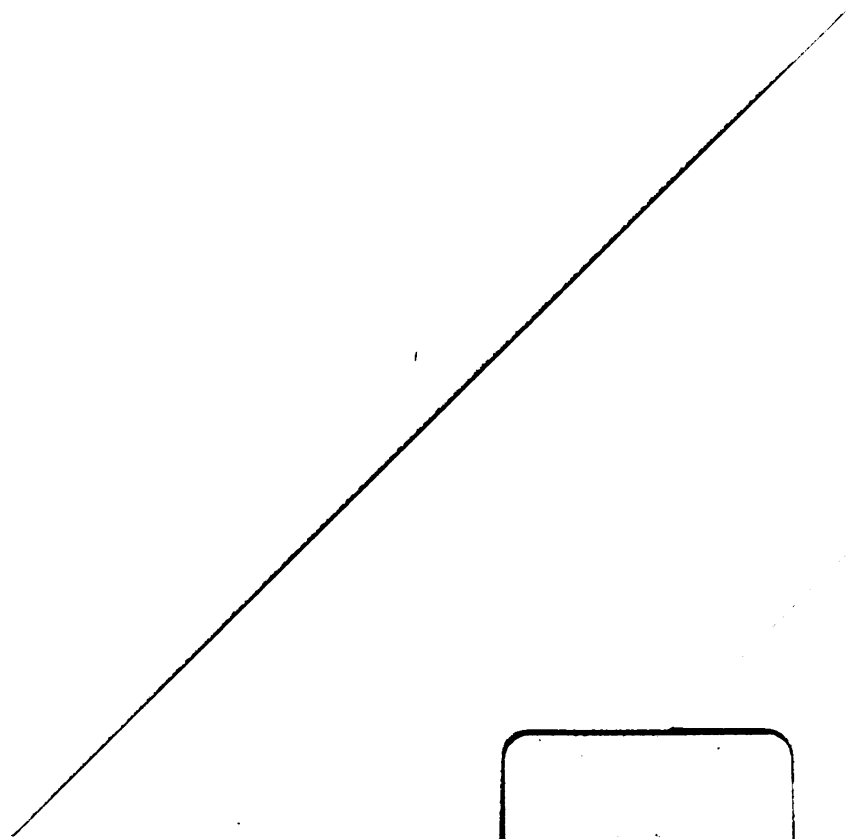
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